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DUNDEE PRECIOUS METALS INC.

NI 43-101 Technical Report

Shahumyan Project, Kapan, Republic of Armenia

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And

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And

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And

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And

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For:
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Approved:

Galen White

"signed and sealed"

Galen White
Managing Director - UK

Certificate of Qualified Person – Galen White

As a Qualified Person of this Technical Report on the Shahumyan Project of Dundee Precious Metals Inc. in Kapan, Republic of Armenia, I, Galen White do hereby certify that:

1) I am a Director and Principal Consultant of CSA Global (UK) Ltd, and carried out this assignment for CSA Global (UK) Ltd, 2 Peel House, Barttelot Road, Horsham, West Sussex, RH12 1DE, UK Telephone +44 1403 255 969, e-mail: galen.white@csaglobal.com

2) The Technical Report to which this certificate applies is titled "NI 43-101 Technical Report – Shahumyan Project, Kapan, Republic of Armenia" and is dated effective 30 September 2014.

3) I hold a BSc (Hons) degree in Geology from the University of Portsmouth, England (1996) and am a registered Fellow in good standing of the Australasian Institute of Mining and Metallurgy (FAusIMM Membership Number 226041) and a Fellow of the Geological Society, London (Member Number 1003505). I am familiar with NI 43-101 and, by reason of education, experience in exploration, evaluation and mining of vein hosted mineral deposits in Europe and Australia, and professional registration; I fulfil the requirements of a Qualified Person as defined in NI 43-101. My experience includes 16 years in mineral exploration and resource evaluation.

4) I visited the project that is the subject of this Technical Report, between 5 and 7 February and 29 and 30 May 2013 for a combined total of 6 days.

5) I am responsible for the following sections of this Technical Report; Sections 1, 2.1, 2.2, 2.3, 2.4.1, 2.4.3, 2.5, 2.6, 3-12, 23 and 25-27.

6) I am independent of the issuer as described in Section 1.5 of NI 43-101.

7) I have had prior involvement with the property that is the subject of this Technical Report, being the preparation of an NI 43-101 Technical Report titled "NI 43-101 Technical Report – Shahumyan Project, Kapan, Republic of Armenia" dated 29 July 2013.

8) I have read NI 43-101 and the parts of the Technical Report I am responsible for have been prepared in compliance with N1 43-101.

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of September 2014.

Galen White

"signed and sealed"

Galen White BSc (Hons), FAusIMM, FGS
Director and Principal Geologist
CSA Global (UK) Ltd.

Certificate of Qualified Person – Malcolm Titley

As a Qualified Person of this Technical Report on the Shahumyan Project of Dundee Precious Metals Inc. in Kapan, Republic of Armenia, I, Malcolm Titley do hereby certify that:

1) I am a Principal Consultant of CSA Global (UK) Ltd, and carried out this assignment for CSA Global (UK) Ltd, 2 Peel House, Barttelot Road, Horsham, West Sussex, RH12 1DE, UK Telephone +44 1403 255 969, e-mail: Malcolm.titley@csaglobal.com.
2) The Technical Report to which this certificate applies is titled "NI 43-101 Technical Report – Shahumyan Project, Kapan, Republic of Armenia" and is dated effective 30 September 2014.
3) I hold a BSc degree in Geology and Chemistry from the University of Cape Town (1979) and am a registered Member in good standing of the Australian Institute of Geologists (AIG Membership Number 2546). I am familiar with NI 43-101 and, by reason of education, experience in the exploration, evaluation and mining of vein hosted mineral deposits in Europe, Australia and Africa, and professional registration; I fulfil the requirements of a Qualified Person as defined in NI 43-101. My experience includes 33 years in mining and resource evaluation.
4) I have visited the project that is the subject of this Technical Report, for a period of three days between 19 and 22 August 2013.
5) I am responsible for Section 14 "Mineral Resource Estimates" of this Technical Report.
6) I am independent of the issuer as described in Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of this Technical Report, being the preparation of an NI 43-101 Technical Report titled "NI 43-101 Technical Report – Shahumyan Project, Kapan, Republic of Armenia" dated 29 July 2013.
8) I have read NI 43-101 and the part of the Technical Report I am responsible for has been prepared in compliance with N1 43-101.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the part of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated this 30th day of September 2014.

Malcolm Titley

"signed and sealed"

Malcolm Titley BSc, MAIG
Principal Geologist
CSA Global (UK) Ltd.

Certificate of Qualified Person – Julian Bennett

As a Qualified Person of this Technical Report on the Shahumyan Project of Dundee Precious Metals Inc. in Kapan, Republic of Armenia, I, Julian Bennett do hereby certify that:

1) I am an Independent Mining Consultant to CSA Global (UK) Ltd, and carried out this assignment for CSA Global (UK) Ltd, 2 Peel House, Barttelot Road, Horsham, West Sussex, RH12 1DE, UK Telephone +44 1403 255 969.
2) The Technical Report to which this certificate applies is titled "NI 43-101 Technical Report – Shahumyan Project, Kapan, Republic of Armenia" and is dated effective 30 September 2014.
3) I hold a BSc degree in Mining from the Royal School of Mines, London. I am a Fellow of the UK Institute of Materials, Minerals and Mining and am a Chartered Engineer. I am familiar with NI 43-101 and, by reason of education, experience in the evaluation and mining of vein hosted mineral deposits in Europe, Africa, Asia, Australia and the Americas, and professional registration; I fulfil the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 48 years in the mining industry.
4) I have visited the project that is the subject of this Technical Report, between 25 and 28 June 2013 for a total of 4 days, and again from 27 and 30 January 2014 for a total of 4 additional days.
5) I am responsible for the following sections of this Technical Report; Sections 2.4.2, 15, 16 and 18.
6) I am independent of the issuer as described in Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of this Technical Report, being the preparation of an NI 43-101 Technical Report titled "NI 43-101 Technical Report – Shahumyan Project, Kapan, Republic of Armenia" dated 29 July 2013.
8) I have read NI 43-101 and the parts of the Technical Report I am responsible for have been prepared in compliance with NI 43-101.
9) As of the date of the Technical Report to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated this 30th day of September 2014.

Julian Bennett

"signed and sealed"

Julian Bennett BSc, ARSM, FIMMM, CEng
Mining Consultant to CSA Global (UK) Ltd

Certificate of Qualified Person – Simon Meik

As a Qualified Person of this Technical Report on the Shahumyan Project of Dundee Precious Metals Inc. in Kapan, Republic of Armenia, I, Simon Meik do hereby certify that:

1) I hold the position of Corporate Director, Processing, of Dundee Precious Metals Inc. the parent company of Dundee Precious Metals Kapan, 4 Gortsavanayin Str, 3301 Kapan, Armenia.
2) The Technical Report to which this certificate applies is titled "NI 43-101 Technical Report – Shahumyan Project, Kapan, Republic of Armenia" and is dated effective 30 September 2014.
3) I hold a BSc degree and PhD in Minerals Engineering from the University of Birmingham, UK. I am a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy-FAusIMM (CP), Membership Number 106146). I am familiar with NI 43-101 and, by reason of education, experience in laboratory research, pilot plant testing, mineral processing plant design, feasibility studies, plant commissioning and study/project/plant operations management in many aspects of small and large mineral processing plants; I fulfil the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 35 years in the mining industry.
4) I have visited the project site that is the subject of this Technical Report on a regular basis since 2006, most recently between 11 and 19 January 2013.
5) I am responsible for the following sections of this Technical Report; Sections 2.4.4, 13, 17, 19-22 and 24.
6) I am not independent of the issuer as described in Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of this Technical Report as Vice President – Processing and previously as Operations Manager of Dundee Precious Metals Inc. My involvement with the project since 2006 has included supervision of all metallurgical testwork, process development and overview of plant engineering.
8) I have read NI 43-101 and the parts of the Technical Report I am responsible for have been prepared in compliance with NI 43-101.
9) As of the date of this Technical Report to the best of my knowledge, information and belief, the parts of Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated this 30th day of September.

Simon Meik

"signed and sealed"

Simon Meik BSc (Hons), PhD, FAusIMM (CP)
Vice President – Processing, Dundee Precious Metals Inc.

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Glossary

%	Percent
°C	Celsius degrees
µm	Micrometer, or 0.000001 m
3D	Three-dimensional model or data
Ag	Silver
Andesite	An extrusive igneous, volcanic rock, of intermediate composition, with aphanitic to porphyritic texture.
Anticline	A fold that is convex up and has its oldest beds at its core
Ar	Argon
Argillic	Hydrothermal alteration which introduces clay minerals including kaolinite, smectite and illite
ASTER	Advanced Spaceborne Thermal Emission and Reflection Radiometer
Au	Gold grade measured in parts per million
AuEq	Gold Equivalent, a grade based on the contribution of other mineral credits, e.g. Cu, Zn, Ag. Pb
Azimuth	An angular measurement in a spherical coordinate system, i.e. deviation degree relative to north
BLEG	Bulk Leach Extractable Gold method of laboratory analysis
Bond W	Bond Rod Mill Index Metallurgical Tests
Bornite	A copper sulphide mineral with a chemical composition of Cu_5FeS_4
BQ	A diamond drill core diameter of 60 mm (outside of bit) and 36.5 mm (inside of bit)
Breccia	A rock composed of broken fragments of minerals or rock cemented together by a fine-grained matrix
CAPEX	Capital Costs
Chalcopyrite	A copper iron sulphide mineral with chemical composition $CuFeS_2$
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetre
CMS	Cavity Monitoring System
Coefficient of variation (CV)	In statistics, the normalized variation value in a sample population
Collar	Geographical coordinates of the collar of a drill hole or a working portal
Compositing	The process of dividing or adding sample intervals together to form a regular length
CompSE	A process in datamine that composites drill hole sample data to honour a defined minimum interval length at a defined minimum grade
Cretaceous	A geological period from 145 Ma to 66 Ma
CRM	Certified Reference Materials
CSA	CSA Global (UK) Ltd
Cu	Total copper grade as a % of the sample mass, sometimes written as TCu
Cut-off grade	The threshold value in exploration and geological resources estimation above which mineralised material is selectively processed or estimated

Datamine	A 3D mining software package
DD	Diamond core drilling method
DGPS	Differential Global Position System
DIP	The angle of drilling (or of a structure) relative to horizontal
DPM	Dundee Precious Metals
DPMK	Dundee Precious Metals Kapan
DTM	Digital Terrain Model. Three-dimensional wireframe surface model, for example, topography
Dynamic Anisotropy	A method which uses a 3D plane of varying strike and dip to control the direction of a search ellipse
Easting	Coordinate axis (X) for metre based Projection, typically UTM. Refers specifically to metres east of a reference point (0,0)
g	gram
g/t	Grams per tonne
Gabbro	A large group of dark, coarse-grained, intrusive mafic igneous rocks chemically equivalent to basalt
Galena	A natural mineral form of lead(II) sulphide with a chemical composition of PbS
GEMCOM	A 3D mining software package
Geochemical sampling	In exploration, the main method of sampling for determination of presence of mineralisation. A geochemical sample usually unites fragments of rock chipped with a hammer from drill hole core at a specific interval
GIS	Geographical Information System
GK8	Gauss Kruger – Pulkovo 1942 Zone 8 projection (“GK8”)
GKZ	Russian Federal Government State Commission on Mineral Reserves
GPS	Global Positioning System
Histogram	Diagrammatic representation of data distribution by calculating frequency of occurrence
HQ	A diamond drill core diameter of 96 mm (outside of bit) and 63.5 mm (inside of bit)
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ICP-OES	Inductively Coupled Plasma Optical Emission Spectroscopy
IDW	inverse distance weighting
Isatis	Geovariances' geostatistical software
ISO 17025	International standards and requirements for the competence of testing and calibration laboratories
Jurassic	A geological period from 201 Ma to 145 Ma
K	Potassium
km	kilometre
km^2	Square kilometre
Kriging	Method of interpolating grade using variogram parameters associated with the samples’ spatial distribution. Kriging estimates grades in untested areas (blocks) such that the variogram parameters are used for optimum weighting of known grades. Kriging weights known grades such that variation of the estimation is minimised, and the standard deviation is equal to zero (based on the model)

LHOS	Long hole Open Stoping method of underground mining
m	metre
M	Million
Ma	Million years
Mean	Arithmetic mean
Median	Sample occupying the middle position in a database
MFT	MinnovEX Flotation Metallurgical Tests
MICROMINE	A 3D mining software package
mm	millimetre
MRE	Mineral Resource estimate
Mt	million tonnes
Mtpa	Million tonnes per annum
Neogene	A geological period from 23 Ma to 2.5 Ma
NI 43-101	Canadian National Instrument 43-101 , CIM Definition Standards June 30, 2011
Northing	Coordinate axis (Y) for metre based Projection, typically UTM. Refers specifically to metres north of a reference point (0,0)
NQ	A diamond drill core diameter of 75.7 mm (outside of bit) and 47.6 mm (inside of bit)
Nugget	The typical difference (for an individual domain) in grade between samples taken immediately adjacent to each other
OK	Ordinary Kriging
OPEX	Operating Costs
oz	Troy ounce (31.1034768 grams)
Pb	Lead
Population	In geostatistics, a population formed from grades having identical or similar geostatistical characteristics. Ideally, one given population is characterized by a linear distribution
Porphyry	An igneous rock consisting of large-grained crystals in a fine-grained matrix
ppm	Parts per million
PQ	A diamond drill core diameter of 122.6 mm (outside of bit) and 85 mm (inside of bit)
Pyrite	An iron sulphide with the chemical composition of FeS2.
QA/QC	Quality Assurance Quality Control
Qp	Qualified Person as defined in NI 43-101
RA	Republic of Armenia
RC	Reverse circulation drilling method
Reserves	Mineable geological resources
Resources	Geological resources (both mineable and un-mineable)
RL	Elevation of the collar of a drill hole, a trench or a pit bench above the sea level
ROM	Run of Mine
Sample	Specimen with analytically determined grade values for the components being studied
SD	Surface Diamond drilling method, completed by KPMK
SEDAR	Online report filing system for the Canadian Securities Administrators
SGS	International laboratory group

Sill	Variation value at which a variogram reaches a plateau
SOP	Standard Operating Procedure
Sphalerite	The chief zinc sulphide mineral with a chemical composition of ZnS
SPI	Sag Power Index Metallurgical Tests
SQL	Structured Query Language, is a special purpose programming language designed for managing data
SRTM	Shuttle Radar Terrain Model
Stockworks	A interconnected network of veins
SURPAC	A 3D mining software package
Swath plot	A method of block model validation using a graph that compares input grades, drill metres, block model tonnes
t	Tonnes
TMF	Tailings Management Facility
Top cut	A value to which anomalously high grades are restricted to, determined by statistical methods
TSNIGRI	Institute of Geology of Armenia
TRUETHK	TRUETHK is a command/procedure in Datamine that allows for the calculation of true thickness of an intercept
UD	Underground Diamond
UDGC	Underground Grade Control
UHC	Underground Historical Channel
UHD	Underground Historical Diamond
USD	United States Dollar ($)
UTM	Universal Transverse Mercator
Variation	In statistics, the measure of dispersion around the mean value of a data set
Variogram	Graph showing variability of an element by increasing spacing between samples
Variography	The process of constructing a variogram
Vein	A sheetlike body of crystallized minerals intruded into a host rock
VMS	volcanogenic massive sulphide
VTEM	Versatile Time Domain Electromagnetic
WGS84	World Geodetic System initialised in 1984
X	The direction aligned with the x-axis of a coordinate system
Y	The direction aligned with the y-axis of a coordinate system
Z	The direction aligned with the z-axis of a coordinate system
Zn	Zinc

1 Executive Summary

1.1 Introduction

CSA Global (UK) Ltd ("CSA") were requested by Dundee Precious Metals Inc. ("DPM") through its 100% owned subsidiary, Dundee Precious Metals Kapan ("DPMK"), to prepare an independent Mineral Resource Estimate ("MRE") for their underground gold and polymetallic Shahumyan mine, located at Kapan in Armenia.

The MRE for the Shahumyan Deposit is presented in Table 1. The cut-off grade was selected in light of the economic sensitivities of the project and informed by current mining practises.

The MRE is reported as at 31st December 2013 which reflects the data used in the estimate and the mined wireframes provided to CSA by DPMK used to deplete the resource.

Following the preparation of the MRE in 2013, DPM began the preparation of a preliminary economic assessment to evaluate a life of mine scenario based on an expanded rate of 1 Mtpa throughput, the results of which are included in this report.

1.2 Project Description and Location

The Kapan mining area is located in the south eastern corner of Armenia at latitude and longitude 39°12'26"N 46° 24'17"E, 320 km south of the capital city of Yerevan. Shahumyan is situated within part of the Tethyan tectonic belt which extends from Southeast Asia to Europe.

The ground held by DPMK in the Kapan area comprises one Mining Licence and one Exploration Licence. The Kapan Exploration License covers 90.7 km^2 with some 12.5 km^2 excised to cover existing populated areas, mine concession, and related infrastructure.

DPMK has operated the Shahumyan Mine concessions under a special mining licence in effect since 2006. In an agreement reached in November 2012, DPMK's Shahumyan mine licence has been renewed under the new Mining Code adopted in late 2011. The license term remained unchanged until 2020. DPMK has submitted an application for license extension up to LoM term as per the new code and currently is awaiting a response from the ministry.

All the required environmental licenses and permits are currently attained and up to date.

Royalty is now calculated based on the following formulae:

Royalty rate = 4% + 100 x (Profit /Revenue x 8)

Royalty = Gross sales x royalty rate

There are no known environmental liabilities to which the project is subject.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Exploration Licence is centred on the town of Kapan which has about 40,000 inhabitants. The town of Kapan itself is situated 320 km from the capital of Armenia, Yerevan. Amenities include:

- The main asphalt road through Kapan is the single main N-S transport route used for freight in and out of Armenia.
- There is an airport with a paved runway at Kapan however it has not been functioning since the Armenian Azeri war (ceasefire 1994).
- Electrification of the Kapan region is provided via connection with the United Energy of Transcaucasia system and provides a relatively uninterrupted supply.
- Both potable and irrigation water are readily available throughout Kapan for industrial use and drilling purposes.
- Mining personnel are sourced locally and there are non-Armenian Nationals employed at the operation.

The region represents a typical mountainous area characterized by split relief up to 3200 m, and overall altitude of 700 m to 3200 m ASL.

The climate of the region is continental with a long lasting warm to hot summer, which is often dry, and short mild winters often with moderate amounts of snow. The field season is generally active between March and November and at other times much of the concession is snow covered.

1.4 History

Historical industrial scale exploitation of the Shahumyan deposit can be summarised as follows:

- Soviet state owned “Shahumyan Mine Group”: 1923 - early 1950’s
- Russian Federal Government State Commission: until 1990
- “Kapan Ore Mining and Processing Enterprise”: 1995 - 2004
- “Kapan Ore Mining and Processing Enterprise” was privatised in 2004
- Renamed to “Deno Gold Mining Company” (DGMC) and operated under an entity named Vatrin until DPM acquisition: 2004-August 2006
- DPM acquired 80% of Vatrin Investments Ltd. (Vatrin), which held 100% of DGMC: August 2006
- DPM acquired 100% of Vatrin by 2010.

From the recommencement of production in 1994 until DPM’s initial acquisition approximately 1,600,000 tonnes of material were extracted at a grade of 1.8 g/t Au, 30 g/t Ag, 0.27% Cu, 0.24% Pb and 1.4% Zn.

Since 2007, DPMK has also undertaken regional as well as focused geochemical and geophysical exploration and structural mapping programs across their licences.

1.5 Geological Setting

In summary:

- The Kapan project is located within the relatively under-explored Lesser Caucasus, part of the extensive Tethyan orogenic belt (approx. 150 Ma). Several metallogenic events have been distinguished in the Lesser Caucasus.
- The Shahumyan mineralised field is thought to represent a large hydrothermal sulphidation system that was emplaced in connection with the intrusion of an underlying felsic intrusive body. The middle Jurassic host rocks comprises of a suite of intermediate tuffs, flows, breccias and sub-volcanic intrusions.
- Mineralisation is hosted by a suite of sub-parallel veins, the current interpretation of the deposit includes in excess of a hundred veins. In many cases the continuity of these veins is yet to be established and future drilling will be important to validate the current model.
- Middle Jurassic to Lower Cretaceous ages of mineralisation were established by crosscutting relationships and by K-Ar dating of altered host rocks in the Alaverdi, Drmbon, and Shahumyan deposits.
- There are flaws or omissions in the existing interpreted structural architecture of the deposit (Davis,2006). Additionally previous mapping campaigns by historical Soviet operators and more recently by external contractors appear to have conflicting models (Tate,2012). Dundee is endeavouring to establish a robust geological model.

1.6 Mineralisation

Mineralisation within the Shahumyan deposit is characterized by:

- Narrow (0.2 – 2.0 m), steeply dipping and approximately east-west striking vein sets
- Veins are typically polymetallic (Cu-Zn-Pb-Au-Ag)
- Veins are often surrounded by a wide zone of argillic alteration and disseminated mineralisation up to 10 – 25 m wide.

Mineralisation species are dominated by:

- Semi-massive to massive sphalerite, pyrite, chalcopyrite, galena and bornite

Accessory mineralisation assemblages include:

- Tennantite, tetrahedrite, bornite, enargite, chalcocite, covellite, Pb, Au, Ag, Bi tellurides and native gold and silver (recorded during petrographic analysis).

Gangue minerals are:

- Quartz, calcite, gypsum, anhydrite and rhodochrosite.

1.7 Exploration

Within a 15 km radius of Kapan numerous showings all related to the Centralni (Cu) and Shahumyan (Au-Ag-Cu-Pb-Zn) hydrothermal systems exist. DPMK is currently systematically evaluating all prospects within the near mine area and have undertaken:

- Regional as well as focused geochemical stream and soil sampling programs
- Geophysical exploration programs employing
    - Various satellite image packages have been acquired by DPMK. These have been used for work planning to assist in target selection and for developing maps.
    - A helicopter borne Versatile Time Domain Electromagnetic ("VTEM") survey, used to assist in the outlining of conductive features such as massive sulphide mineralisation and wide zones of alteration or graphitic units up to 400 m depth.
- Detailed structural mapping exercises, often employing international consultancies to undertake high level interpretations.

CSA believes that appropriate exploration methods and QA/QC programs have been employed by DPMK, however in-depth review of this work falls outside of the scope of this report.

1.8 Drilling and Surveying

The database consists of both DPMK and Historical drilling data.

1.8.1 DPMK Drilling

DPMK has carried out surface diamond drilling and RC drilling since July 2007. A total of 598 diamond drill holes and 102 RC drill holes have been completed for 181,309 m and 15,132 m respectively. DPMK has also undertaken approximately 176 underground diamond holes for a total of 57,802 m.

Historical drill data in the database includes 1,501 underground diamond drill holes totalling 277,025 m and 6,750 m of undifferentiated underground grade control data.

The available data, both historical and DPMK (and including channel sample and grade control data) is summarised in Section 10.

DPMK drill data collected includes RQD, core and RC recoveries, core photography and geological logging, typically using personal Tablet computers running Field Marshall software; an acQuire interface.

CSA considers the procedure described above regarding data collection methods as adequate; however a review of the dataset showed that there is insufficient detail in the logging to classify mineralisation based on lithcode, and improvements are required. This has been discussed in more detail in Section 12.2.

1.8.2 Pre-DPMK Drilling

Historical data (including drill, grade control and channel sample data) has been compiled from four principal sources. The main types of data captured were:

- Drill hole logs from historic resource definition drilling from the 1960's until 1990's
- 1:1000 scale cross sections
- 1:200 scale sample plans with detailed structural mapping
- 1:1000 level summary plans with development and survey stations
- Vein Projections detailing historic resource calculations.

All historical hard copy data was entered manually into data entry software and then imported into acQuire database management software. Graphical documentation was captured digitally using a GTCO-Calcomp roll-up digitiser system and scanned using a Colortrac CX-40 scanner. Validation of historical data included cross checking collar positions and log assay results with plans and sections.

Core descriptions recorded lithology, texture, alteration and mineralisation style however historic lithological logging was descriptive and not standardised and therefore currently no geologic descriptions have been translated and entered into a database.

1.8.3 DPMK Surveying

In summary:

- Surface collar surveying utilises a GPS Topcon HIPER GA tools with an accuracy of 3 mm - 10 mm.
- Underground collar surveying utilises a Leica TC 15a total station setup and an Optech Cavity Monitoring System (CMS) for void modelling, with a small Ranger Survey tool for long hole stoping drill hole checking.
- Surface down-hole surveying is undertaken by gyroscope, manufactured by Stockholm Precision Tools AB of Sweden. This tool is regularly calibrated and error is ±1% for azimuth.
- RC holes are surveyed either within rods or within open hole at 25 m intervals. Diamond holes are surveyed at 25 m in open holes or at 50 m intervals if surveyed during rod pulls.
- Underground drill holes are surveyed every 30 m during rod pulls using the Reflex EZ-Trac camera system.

- Production drill holes are down-hole surveyed using a Ranger Mask 2 multi/single shot system, DPMK reports that 20% of drilled production holes are surveyed.

CSA considers this procedure to be appropriate.

1.8.4 Pre-DPMK Surveying

- Hole surveys were likely undertaken using optical methods; using theodolites and survey loops.
- Collar locations for historical holes were ultimately digitised from underground hard copy plans. Collars and traces were desurveyed in a mining package and plotted positions checked back against the original plans.
- DPMK located 563 hardcopy downhole survey reports in various conditions for the approx. 1,500 underground historic drillholes in the Shahumyan deposit. CSA believe that the new downhole survey data is an improvement relative to the previous dataset used in the MRE, although lack of initial azimuths at 0 m, lack of regular shots downhole and incomplete datasets are not an ideal.

1.9 Sampling and Analysis

1.9.1 DPMK Drill Sampling

Factors undertaken to address contamination and ensure representivity for surface Diamond (SD), Underground Diamond (UD) and surface RC drilling (RC) and are:

- Half core samples were collected
- RC samples were passed through a riffle splitter and cleaned between each sample using a rubber hammer and compressed air.
- Damp or clayey RC samples were split using a polypipe spear.
- The RC cyclone is checked for sample build up during each rod pull (6 m).
- RC sample weights were routinely recorded, to monitor sample recovery.
- RC drill is converted to Diamond whenever unmanageable water is encountered.

CSA considers the drilling procedures to be appropriate.

1.9.2 DPMK Channel Sampling

Concern was raised in the previous MRE (CSA,2013[2]) regarding Underground Channel (UC) sampling procedures at Kapan. These procedures were subsequently reviewed onsite by CSA Principle Mine Geologist Steve Rose (CSA,2014) and all concerns by CSA have been addressed. UC sampling procedure includes:

- The face location measured using either a Disto instrument or a tape measure, using a survey wall station as control.
- Samples are taken using a Hilti hammer drill fitted with a chisel blade over the width of the panel (10 cm) and along the entire mapped face.
- Sample weights are about 4 kg in size

CSA considers the channel sampling procedure to be appropriate.

1.9.3 DPMK Sample Preparation

Sample preparation is undertaken at the SGS laboratory on site using laboratory preparation procedures which are considered by DPM and CSA to be appropriate for this style of mineralisation.

The majority of DPMK analyses are completed at SGS Kapan and SGS Chelopech. The laboratories do not have full ISO 17025 accreditation but do apply ISO approved procedures and management certificates. SGS facilities in Western Australia and Canada carry full ISO 17025 accreditation. The facilities are all independent of the issuer.

Analytical methods at SGS Chelopech are detailed below:

- Gold: was analysed by either a 25 g or 50 g fire assay with AAS finish.
- Silver, lead, zinc, copper, arsenic, mercury and bismuth were analysed by 0.3 g aqua regia digest with AAS finish for high grade samples (Chelopech) with low grade samples analysed by ICP-OES (Tianjin and Welshpool).
- For each batch of 40 samples the laboratory inserted 4 replicates, 3 second splits, 3 CRMs and a blank sample, for QA/QC purposes.

1.9.4 DPMK QA/QC

A comprehensive quality assurance and quality control programme has been implemented by DPMK for surface and underground drilling with samples being analysed at the SGS managed Kapan onsite laboratory. No DPMK QA/QC data was available for the underground channel samples or the historical data, but the laboratory inserted QA/QC material for the underground channel samples was available.

QA/QC results for gold, copper and zinc for the DPMK drilling (DPMK and laboratory material) and underground channel sampling (laboratory material only) were reviewed. No fatal flaws were apparent.

1.9.5 DPMK Security of Samples

All samples are securely transported by DPMK Staff from the drill rigs to the sample preparation facility. The SGS managed analytical laboratory is located within the confines of the DPMK compound, access to which is secured by locked gate and 24 hour guard. Samples sent to SGS Chelopech (Bulgaria), SGS Tianjin (China) or SGS Welshpool (Australia) for final analysis are packed into sealed cardboard boxes.

1.9.6 Pre-DPMK Sampling

All Pre-DPMK sampling information regarding Drill, Underground Grade Control and Underground Grade Control Sampling has been gathered from hard-copy reports and communications with previous staff.

- Underground Diamond Drill (UHD) sampling intervals typically honoured vein or lithological contacts and extended sampling 4 m either side of such a mineralised zone. However, there appears to be variation in sampling procedures within the historic dataset during different periods of exploration. All core storage facilities in Shahumyan have been destroyed and no useable core exists.
- Underground Grade Control (UDGC) sampling is classified as undifferentiated however it is essentially short, diamond drill, core holes of max. 1.5 m length following equivalent split core sampling procedures as for UHD samples.
- Underground Historical Channel (UHC) sampling was undertaken at 4 m intervals for every second face advance with crosscuts routinely sampled at 1 m intervals. All sampling honoured geological boundaries with sub-samples. Samples were collected using a hammer and chisel while standing on a sheet of fabric to catch the sample to be placed in calico bags. Minimum sample weights were 4 kg to 7 kgs.

1.9.7 Pre-DPMK Sample Preparation

Generally all samples were analysed for Cu, Au, Zn, Ag and Pb. Between 1972 and 1976, scientific research institutions from Moscow also conducted limited core assay for S, Fe, Se, Te, Cd, Ga and In. Typically samples were sent to various independent laboratories within Armenia and the greater Caucuses area for assay. A summary of assay methods include:

- 1962-1994 Cu by Iodometric titration
- 1962-1980 Pb and Zn by Polarographic method
- 1980-1994 Pb and Zn by Iodometric titration
- 1962-1994 Ag + Au by Fire Assay

1.9.8 Pre-DPMK QA/QC

No QA/QC data nor hole orientation data exists for the historical drilling and channel sampling. As a result; CSA is unable to verify the quality of the pre-DPMK underground drill and channel sample data used in this MRE, except to review it relatively against data collected by DPMK. CSA was satisfied by the results of this comparison work.

1.10 Data Verification

1.10.1 Sample Type Review

CSA undertook a detailed review of the sample and assay dataset since the MRE dataset is reliant on samples collected from a variety of methods (Drilling, channels and grade control) both by DPMK and historical operators. These different datasets are intrinsically different w.r.t grade distributions however observations and documentation indicated that appropriate standards were typically met by all drill/sample types and that there is no obvious reason for grade bias. The review was undertaken both statistically and spatially employing:

- Summary statistics
- Probability plots, and histograms
- Swath plots
- And by undertaking Kriged and Nearest Neighbour estimates of larger veins using combinations of the different datasets.

The review indicated a difference in the grade distributions between the different drill/sample types. The differences are attributed to:

- UD and UHD as well as UC and UHC datasets have sampled different portions (i.e. grades) of the Shahumyan resources, during different periods of resource development. There are no areas that allow for a 1:1 comparison
- Drill data have lower global grades as they typically sample larger lengths of surrounding waste.
- Attempts to compare grade distributions using data flagged within the Vein Model was complicated by poor geological logging to constrain mineralisation intercepts and the fact that a large proportion of high grade intercepts fall outside the vein model.
- The above point is further complicated by the fact that mineralisation is nuggetty and metal proportions vary within single veins preventing 1:1 comparisons for different datasets that have intersected the same vein.

Based on the review work undertaken by CSA as well as observations made onsite made by CSA staff, it is believed that all drill and channel sampling data are sufficiently reliable to be included in the estimation of resources.

1.10.2 Geological Logging

CSA's review of logged lithologies showed that it is difficult to establish a correlation between mineralisation and rock type. This is exemplified by:

- The low number of lithology codes within the vein model which indicate mineralisation.

- The prevalence of mineralised intercepts that fall outside of the vein model, which have not been classified with a rock code that indicates mineralisation.

CSA believe that lithological logging should correlate better with mineralisation. Since completion of the work, additional paper logging data has been found which appears to support this observation.

CSA understands that a program of additional data entry and re-logging is currently being undertaken by DPMK in order to address this issue.

1.10.3 Bulk Density

CSA's review work of density data replicated the results of the previous MRE namely that there appears to be poor correlation between the digital logging and the density data. In summary:

- The global dataset SG was 2.67 g/cm3, of this 76% of the data was logged as VAN with an average SG of 2.67 g/cm3
- 70% of all densities >3.9 g/cm3 were logged as VAN
- 80% of the data flagged within the vein model was logged as VAN, with an SG of 2.71 g/cm3
- There is a clear relationship between grade and density.

There is no defined relationship between lithology or location and densities; although it understood that vein material is denser.

CSA believes that there is clearly a relatively broad distribution of densities at Kapan, however current geological logging does not adequately constrain these differences, and therefore has a material impact on the estimation of tonnages.

CSA understands that a more detailed density program is currently being undertaken by DPMK in order to address this issue. As discussed above, the more detailed paper logging data which was found subsequent to the completion of this work, may also improve the correlation of density with lithology.

Due to the relationship between density and grade a polynomial regression was used to estimate density from grade following estimation of the block model, see Section 12.3.

1.10.4 QA/QC Database Validation

The DPMK data was loaded into an SQL database for verification by CSA. This SQL database has constraints and triggers which highlight issues such as duplicated records, overlapping or negative intervals, abnormal values (dips, azimuths, recoveries, RQD), etc. The DPMK drilling data (surface and underground) are comprehensive, of a high standard and no significant issues were noted. Less data were available for the historical and channel datasets, but no fatal flaws were observed.

A QA/QC report for blanks, Certified Reference Materials (CRMs), field duplicates and lab pulp splits was produced. No fatal flaws were observed.

1.10.5 Site Visit

CSA Principal Geologist Mr Galen White made a site visit to Kapan in February 2013 (CSA,2013[1]) as QP.

CSA Principal Consultant Mr Malcolm Titley made a site visit to Kapan between 19th and 22nd August 2013. The site visit included a review of the resource estimation work being undertaken by site geologists.

Independent Mining Consultant to CSA, Mr Julian Bennett has made two site visits to Kapan; the first between 25 June and 28 June, 2013, and the second between January 27 and 30 2014, for the purposes of reviewing current mining activity, practises, equipment and facilities.

1.11 Mineral Processing and Metallurgical Testing

External metallurgical test work at Kapan was undertaken for Deno Gold by CSA in 2003, by SGS as part of DPM's Due Diligence review program in 2006, and more recently by SGS in 2013 in both Cornwall and Lakefield.

SGS Minerals (2006) under took grinding and flotation testwork which confirmed the potential of the existing circuit, and which was eventually achieved after a limited modernization program in the plant over the following years of operation.

More recent work has examined various aspects of current plant performance, as well as the characterisation of materials present in potential open pit expansions.

1.12 Mineral Resource Estimates

CSA received the database export from DPMK on 4 October 2013. This data comprised 15,932 holes and channels for 576,335 m.

Wireframes representing vein mineralisation were digitised by DPMK geologists during 2013, and were interpreted using a nominal cut-off of 1.5 g/t Au Equivalent ("AuEq") at a minimum downhole length of 1 m. This includes vein and selvedge mineralisation but do not take into account a minimum mining width, which is 1.8 m at Shahumyan using Long Hole Open Stoping method ("LHOS").

Using the wireframes that DPMK provided, a process of creating mineralisation wireframes honouring the minimum mining width was completed by CSA. These 'diluted' wireframes were used in the MRE to model the volume of vein related mineralisation.

The dip and dip directions of modelled veins were estimated into the block model in order to use Dynamic Anisotropy to inform the estimation search ellipse. This allows localised search rotations during estimation.

Samples were composited downhole to lengths equal to the minimum mining width of 1.8 m based on vein and drill hole / channel orientation. This was required to ensure equivalent sample support for grade estimation using Ordinary Kriging. The dominant sampling interval is 1 m for drillholes. Sample lengths of channels are more variable because sampling honours geological boundaries.

The volume block model was created in Datamine. A parent block size of 3 m x 3 m x 3 m (X x Y x Z) was selected for grade estimation. Sub-celling down to 0.5 m x 0.5 m x 1 m was used to honour mineralisation and mined volumes. The block models were regularised to 3 m x 3 m x 3 m and imported into ISATIS for grade estimation.

Ordinary Kriging was used to estimate grade for Cu, Zn, Pb and S into the 3 m x 3 m x 3 m vein block model. The grades for Au and Ag were estimated using the Top-Cut Model for Deposits with Heavy-Tailed Grade Distribution. This method attempts to deal with high grade outliers in a way that reduces their influence, but does not remove metal from the resource (see Section 14.4.5 for more detail on this method).

Soft boundaries were used between mineralised veins, but the search ellipse was constrained to reduce undue influence of samples from other veins in areas of good sample support.

Following completion of the estimate, AuEq was estimated using the following formula provided by DPMK in October 2013:

$$AuEq=Au+(Cu\times1.2)+(Ag\times0.02)+(Zn\times0.34)$$

In situ dry bulk density was assigned to blocks using a 4^{th} order polynomial regression based on the relationship between AuEq grade and density (Section 1.10). The regression is:

$$(y) = EXP\ [((x^4)*\ 00026921)+((x^3)*\ 00253321)+((x^2)*\ 00305893)+(x*\ 01235032)+00333878]$$

Where y = calculated density and x=grade (Au Eq)

Validation of the MRE consisted of:

- Visual check of cross sections at a local scale comparing model and sample grades.
- Comparison on a global scale; of grade histograms and probability plots
- Swath plots
- Checks on the individual mean grades of the larger veins
- Validation of the estimated in-situ dry bulk density with block grade

Classification was based on a number of criteria, including assessment of the reliability of geological, sample and drill data, and location of production. CSA concluded that the MRE model reflected the tenor of input grade, which can be considered reliable on a local scale in the Indicated areas, and reliable on a global scale in the Inferred areas. Measured Mineral Resources have not yet been classified at Shahumyan as it is the opinion of the QP, that reconciliation of the mineral resource against production data and the grade control model is required before there is a possibility of classifying any part of the mineral resource as Measured.

Mineral Resources estimated for Shahumyan and detailed in this Technical Report are prepared in accordance with the CIM Definition Standards June 30, 2011 and Section 5.1 of National Instrument 43-101 – Standards of Disclosure for Mineral Projects, Form 43-101F1 and Companion Policy 43-101CP.

Table 1. Classified Mineral Resource Estimate for Shahumyan Deposit

Dundee Precious Metals - Kapan. Shahumyan Mineral Resource Estimate as at 31st December, 2013													
Resource Category	MTonnes	**Grades**							**Metal Content**			
		AuEq* (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	S (%)	Au Moz	Ag MOz	Cu MLbs	Zn Mtonnes
Indicated	3.0	5.33	3.21	49.88	0.44	1.77	0.19	2.08	0.3120	4.8480	29	0.0535
Inferred	9.5	4.62	2.83	43.34	0.43	1.21	0.10	3.13	0.8626	13.2100	90	0.1147

MRE is reported using a gold equivalent cut-off of 2.24g/t and an operating net profit cut-off of > USD 0
tonnages are rounded to the nearest 1,000 tonnes to reflect this as an estimate
metal content is rounded to the nearest 100 tonnes or 100ozs to reflect this as an estimate
**AuEq = AuEQ [Au + Cu*1.20 + Ag*0.02 + Zn*0.34]*

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The Mineral Resource has been classified as Indicated and Inferred in compliance with Section 1.2 - Rules and Policies NI 43-101. There are currently no defined Mineral Reserves for the Project.

To the best knowledge of CSA, the stated mineral resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues that prevent this mineral resource from reasonable prospects for economic extraction.

The Kapan MRE is reported at a conceptual profitability cut-off greater than USD 0 and a gold equivalent cut-off of 2.24 g/t. The profitability value is calculated by a set of formulae, and represents a conceptual profitability of each individual block, when considerations such as metal price and current recovery are taken into account, derived from operational experience. It should be noted that the input parameters used in this formula have not been generated following any formal NI 43-101 compliant Feasibility Study, which has not been undertaken, and mineral reserves have not been defined.

The reporting of the mineral resource estimate using both a grade cut-off and the conceptual profitability cut-off has been completed to satisfy the criteria of “reasonable chances of eventual economic extraction” under CIM guidelines.

There has been inadequate analysis at the project to report mineral reserves; however the mine is currently operating and producing saleable product. In addition a Preliminary Economic Assessment ("PEA") has been completed and is disclosed in this Technical Report (see Sections 16-24). A PEA is a study, other than a Pre-feasibility or Feasibility study that includes an economic analysis (see Section 22) of the potential viability of Mineral Resources.

1.13 Mining Operations

1.13.1 Production Rates and Mining Method Review

DPMK's 100% owned Shahumyan underground mine is producing run-of mine material at an average rate of 1,200 tonnes per day (tpd). The mine expansion contemplates a ramp up production over a two year period, to attain an annual production rate of 1 M tonnes. Production and forecast rates for the LOM are present in table 2.

The reader is cautioned that, this information is based on a PEA of the Shahumyan Project, that is preliminary in nature and includes Inferred Mineral Resources, that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and that there is no certainty that the PEA will be realised. No Mineral Reserves have been estimated. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 2. Production and Forecast Rates LOM

LoM	Units	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Development mineralised material	000 t	1,372	177	187	173	214	192	178	166	65	20
Stope mineralised material (tonnes)	000 t	6,620	379	484	718	821	821	821	823	929	826
Total Mineralised Material	000 t	7,992	556	672	892	1,034	1,013	998	989	994	846
Mine Head grade Au	g/t	2.42	2.14	2.59	2.43	2.43	2.39	2.46	2.45	2.41	2.38
Mine Head grade Ag	g/t	37.8	38.5	39.9	37.3	35.8	35.1	34.3	36.9	35.8	46.6
Mine Head grade Cu	%	0.36	0.33	0.33	0.28	0.30	0.31	0.32	0.39	0.35	0.35
Mine Head grade Zn	%	1.05	1.32	1.32	1.16	1.13	1.04	0.85	0.88	0.78	0.95
Mine Head grade Pb	%	0.08	0.08	0.13	0.10	0.10	0.07	0.05	0.06	0.04	0.10
Mine Head grade S	%	2.39	2.72	1.96	1.79	1.96	1.90	2.42	2.55	3.12	3.12
Total Waste	000 t	5,484	386	625	636	640	628	643	636	645	645
Total Material	000 t	13,476	942	1,297	1,528	1,674	1,640	1,641	1,624	1,639	1,491

Access into the mine is via two decline ramps, one to access the south section of the mine, and one to access the north section. Crosscuts are developed from each of the declines. These intersect the veins at approximately 90°. Mine levels are at 15 m vertical intervals.

The capital development (declines and crosscuts) are developed using mechanised electric-hydraulic drill jumbos and vein drives are developed using hand-held pneumatic machinery. In the north section of the mine, capital development is opened using hand held drilling equipment.

Sub-level open stopes are opened up along the vein drives. Each stope is typically 56 m high and 60 m along strike. Production blast holes are drilled upwards from the lower sub-level, and consist typically of parallel holes. All mine development waste is backfilled into each completed stope.

Broken material is loaded in the mining drive by various types of load-haul-dump machines (LHD) directly into diesel haul trucks and hauled to the surface.

The veins containing most of the mineralisation are narrow, down to 0.2 m in width and up to as much as 2.0 m in width (the average width being about 1 m). The MRE is modelled to a minimum width of 1.8 m to included appropriate dilution when required. The minimum stoping width with the sub-level stoping method in use is around 1.8 m, however; due to geometric and mining issues stopes can be wider than 1.8 m resulting in mining dilution. The amount of mining dilution is currently being analysed and procedures reviewed. New drilling machinery is expected that will improve efficiency and accuracy of the stope blasting and improve dilution factors.

1.13.2 Recovery Methods

The plant produces two concentrates, by sequential flotation. Following conventional two-stage crushing and grinding by a rod mill/ball mill circuit, the first is a copper/pyrite concentrate containing gold and silver. Following roughing, scavenging and concentrate re-grinding, cleaning is completed in 3 stages. The scavenger tails are then conditioned prior to zinc rougher and scavenger flotation, with the Zinc scavenger tails being the final process tails. Rougher concentrate is cleaned in 3-stages, with the cleaner tails being recirculated to the previous stage.

The capacity of the current milling circuit is 2,100 tpd while that of the flotation section is in the range of 2,300 tpd. Copper recovery to the final copper concentrate ranges between 85% and 89% to a grade of 20% to 25% Cu. Gold and silver recoveries into the copper concentrate typically range between 70% and 75%, with the grade of gold in the copper concentrate typically between 90 – 110 grammes per tonne (g/t). Zinc recovery to final concentrate ranges between 85 and 92% (depending on head grades) with grades between 58 and 63%Zn. These contain a further 10 – 15% of the gold, bringing overall gold recovery to 80 – 85%.

Copper concentrate, shipped in bulk bags, is transported by truck to Yerevan, and shipped onwards by rail to Poti port in neighbouring Georgia. Zinc concentrates follow the same route, but are shipped in bulk from site, and in containers from Yerevan.

Final tailings are pumped to the current tailings management facility (TMF). This is located in a valley and has a north and south wall. A decant tower reclaims water which flows by gravity back to the mill. Current water usage in the mill consists of 70% reclaim water from the TMF and 30% fresh make up water.

1.13.3 Infrastructure

Power used on site is supplied by the Russian owned “Electric Networks of Armenia” CJSC. Power is stated to not be a constraint at current mine production rates, and is not considered an issue.

Much of the water for the plant and mine is recirculated from the TMF. Make up water is taken from the Voghdji River that flows through the town. There is no evidence of water shortages.

The town of Kapan is on the national road network and is reached by standard vehicles. The site elevation is around 800 m above sea level. There is no direct connection to the national railroad system.

1.13.4 Market Studies and Contracts

Copper and zinc concentrates are currently being sold to several metal traders (including Marc Rich, Louis Dreyfus, and Transamine). At the time of the technical visit, all concentrates were being routed to Chinese customers. No additional market studies have been completed by the issuer.

1.14 Environmental Studies

1.14.1 General

DPMK has been conducting comprehensive baseline environmental studies over the Kapan licence area since 2007. The studies were carried out by its own personnel as well as contracted companies such as Mining and Metallurgical Institute (Yerevan, Armenia), Golder and Associates (Canada), Kingett Mitchell (New Zealand), AATA International (USA), and Geolink Consulting (Russia).

There is no any significant environmental liability other than closure of current operation is currently held by the company.

Environmental fines were paid by the company throughout 2007-2013 and ranged from 3,500 to 30,000 USD. The fines were paid due to disturbance to vegetation and soil, incidents occurred on the tailings pipeline and above limit water discharges.

1.15 Capital and Operating Costs

Table 3 presents the capital expenditure expected for the remainder of the Life of Mine.

The reader is cautioned that this information is based on a PEA of the Shahumyan Project that is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and that there is no certainty that the PEA will be realised. No Mineral Reserves have been estimated. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 3. LOM Capital Expenditure (9 Years)

	USD ('000)
Mine expansion	10,555
Mill expansion	6,134
TMF expansion	13,440
Other growth capital	1,878
Total Initial Capital	**32,007**
Mine sustaining capital	24,854
Mine capital development	120,234
Plant and surface sustaining capital	40,517
Closure	11,995
Total	**229,606**

Operating costs following the expansion, including royalty payments and G&A, are USD 457.5M, equivalent to an average cash cost per tonne of mineralised material of USD 60.40.

Table 4. Operating Cost Summary (LOM)

Item	Unit	LOM
Throughput	Mtpa	0.9
mining cash cost per tonne of material	USD/t	19.36
processing cash cost per tonne of material processed	USD/t	16.84
Services cash cost per tonne material	USD/t	6.88
royalty per tonne of material processed	USD/t	8.57
admin cash cost per tonne processed	USD/t	8.74
Total cash cost per tonne of material processed	USD/t	60.40
Cash cost (by-product)	USD/oz	336

The current operating cost, excluding G&A, is USD 68.00 per tonne of material processed. As production is increased with the throughput ramp-up, significant efficiencies of scale should be realized, as shown in Table 60. The average operating cost, excluding G&A, per tonne of mineralised material over the mine life is expected to be USD 51.66.

1.16 Study Conclusions

The conclusions relevant to this study are:

1.16.1 Procedures

During site visits undertaken by Galen White, meetings were held with the database manager and procedures reviewed in the core yard, exploration department and SGS laboratory. Conclusions based on this site visit included:

- Procedures employed during logging, splitting and sampling of drill material is of an appropriate standard, with the core farm and digital data collection methods well managed.
- The onsite acQuire database is robust and of an appropriate standard however the historical database was not readily verifiable.
- SGS Assay laboratory in Kapan appeared to be well run, no formal audit was completed.

1.16.2 Geological Model

CSA believes that further work is needed to constrain the geological controls on mineralisation at Shahumyan.

The following aspects of the geological model require consideration:

- The current wireframes used in the MRE represent mineralisation, however, what part of the mineralisation is vein and what part is selvedge is uncertain.
- The potential for a discrete and potentially high grade gold mineralisation style has been observed by DPMK staff. This style is currently not being exploited and is characterised by subtle hydrothermal overprinting and argillic alteration (more typical of lode gold deposits). This style is in contrast to the dominant and visible quartz and carbonate veins with massive sulphides currently exploited.
- If this mineralisation exists at Shahumyan, it may explain the relatively poor correlation between the available digital lithology logging and grade. Review of the recently found historical detailed paper logging may assist in improving the relationship between geology logging and mineralisation style.
- Some veins have been modelled dipping to the north. The controls that cause these veins to change dip are not clearly understood. It has also been observed by DPMK that most of the northerly dipping veins have better than average grades and larger than average widths.

1.16.3 Geological Logging

CSA considers that the onsite procedures employed to collect and capture geological observations are appropriate.

However a review of the geological data collected showed a poor correlation between grade and rock type as well as density and rock type. This was demonstrated by:

- The prevalence of mineralised intercepts that fall outside of the vein model which have not been classified with a rock code indicating either a vein or a mineralised structure.
- The dominance of lithologies other than vein that are flagged within the Vein Model (predominantly andesite).

From both a resource as well as a mining perspective it is critical that controls on higher grades and densities are well understood. This has impacts on modelling and statistical domaining in future resource estimates as well as mining operations and development planning.

DPMK are currently undertaking a re-logging exercise which may help to address this issue. This may impact the model which is reliant on geological interpretations to establish continuity between mineralised intercepts. CSA note that since this work was completed additional historical paper logging records have been found. This additional information may assist in resolving the issues discussed above.

1.16.4 Sample Type Review

CSA undertook a detailed review of the sample and assay dataset since the MRE dataset is reliant on samples collected from a variety of methods (Drilling, channels and grade control) both by DPMK and historical operators.

CSA understands that the different sample/drill datasets are different with respect to grade distributions because they are sampling different locations but in summary; observations and documentation regarding drilling and sampling procedures at Kapan and historically, are that appropriate are met by all drill/sample types. Based on this, CSA is of the opinion that there are no obvious reasons for sampling bias and channel samples have been included in this MRE update.

1.16.5 Assay QA/QC

The results for blanks, standards, field duplicates and lab duplicates (repeats and splits) for gold, copper and zinc for the UG, SHA and UGC datasets have been reviewed and summarised below. All samples were analysed at the onsite SGS laboratory.

- CRMs and blanks performed acceptably
- Field duplicates exhibited a significant bias to the original for gold, a lesser bias for Zn and no bias for copper. The mineralised body has a high nugget effect and as only a

few field duplicates have been analysed, no conclusion regarding this bias can be made at this stage.

- Lab repeats have acceptable performance with no bias.

- Issues noted in the previous MRE (January 2013) such as instances of apparent misidentified QA/QC samples and anomalous blank results appear to have been addressed in this review period. Ongoing monitoring is required.

1.16.6 Database Validation

CSA undertook a suite of SQL validation checks on the DPMK database export provided and concluded that:

- The DPMK drilling data (surface and underground) are comprehensive, of a high standard and no significant issues were noted.

- Less data was available for the historical and channel datasets, but no fatal flaws were observed.

1.16.7 Historic Data Verification

A large proportion of the MRE dataset is reliant on historical data. This data has been digitised from hard copy plans and tables. Historical data also lacks QA/QC data as well as verification datasets regarding collar positions and down-hole surveying. As a result:

- CSA is unable to verify the accuracy or validity of the historical data against the more recent data collected by DPMK. In summary the grade populations correlate sufficiently to be included in the MRE.

- CSA understands that hard copy documentation of historic down-hole orientation data has recently been discovered by DPMK and is currently being entered into the database.

- DPMK located 563 hardcopy downhole survey reports in various conditions for the approx. 1,500 underground historic drillholes in the Shahumyan deposit. CSA believe that the new downhole survey data is an improvement relative to the previous dataset used in the MRE, although lack of initial azimuths at 0 m, lack of regular shots downhole and incomplete datasets are not an ideal.

1.16.8 Bulk Density

Conclusions from CSA’s review of the density data include:

- Geological control regarding the distribution of densities at Kapan remains uncertain. This is largely attributable uncertain relationships between density values and geological logging of these samples for samples within and out of the vein model. CSA note that since this work was completed additional historical paper logging records have been found. This additional information may assist in improving the relationship between density and geology logging.

- Mine personnel are of the opinion that densities are higher within the vein than currently indicated by the data review, i.e. by flagging data by the model and/or classifying data based on lithology code.

- A clear trend between grade (Au Eq) and density supports this onsite observation; that mineralised veins are denser. As a result the use of a regression from which to calculate density based on estimated grades is considered appropriate. Based on this relationship a 4th order polynomial was defined and used in this updated estimate of resources resulting in an increase of 1% tonnes at a zero cut off (Au Eq) but with an increase of 1 to 2% tonnes above a 3 g/t Au Eq.

- Although the estimated densities reflect an improvement in modelling the variability of density in the block model, further work is required to improve the regression and to determine what density data is appropriate for use within the regression. This is made clear because the current dataset used for the regression (only data that was flagged by the vein model) does not reflect the grade distribution of the estimated model. Based on this discrepancy in distributions, it appears that the regression may be conservative in its calculation of density at lower grades.

- As additional density data is collected, the density assigned to the model may change and affect the resource tonnage.

1.16.9 Mineral Resource Estimation

158 vein wireframes representing 79 vein clusters were modelled by DPMK geologists, under the supervision of Mr Ross Overall, Resource Geologist DPM. Veins were interpreted using logged geology, underground mapping and drill hole structural data where available. Wireframes were interpreted on a nominal cut-off of 1.5 g/t Au Eq and 1 m down the hole length. Most veins were interpreted to strike approximately E-W/SE-NW and dip moderately to steeply to the south. Some veins were interpreted to dip to the north.

The vein wireframes were provided to CSA and used as the basis to create a set of mineralisation wireframes that honours the minimum mining width achieved at Shahumyan using the Long Hole Open Stoping method of mining. The CompSE process in Datamine software was used to selectively composite data to create the diluted wireframes. The composites produced had to meet a minimum grade of 1.5 g/t Au Eq and a minimum true width of 1.8 m (i.e. downhole lengths varied depending on orientation and dip of drilling, and orientation and dip of veins). 158 mineralisation wireframes were produced using Implicit Modelling and DTM Creation in Micromine software and were used to inform mineralisation volume in the MRE.

In addition to the mineralisation wireframes, a dyke model, development and mining depletion wireframes, and weathering surfaces provided by the client were used to flag data and the model.

Assay data was composited to 1.8 m true width to honour the dominant true width of vein intercepts and stopes. Statistical review of the resultant composites did not indicate bias. Data was composited using vein boundaries to start and stop compositing.

Statistical analysis was completed on the composites. Statistics for Au, Ag, Cu, Zn, Pb and S were reviewed. Au and Ag were characterised by heavy-tailed, skewed grade distributions, while the remaining variables had normal populations most easily interpreted using normal score transforms.

Variography was completed for use in Kriging. Variogram models for Cu, Pb, Zn and S were characterised by moderate to high nuggets for most variables, with two structures modelled. Gaussian transforms were used in the majority of cases and variogram models were back-transformed prior to use in Kriging.

The Top Cut Method for Deposits with Heavy-Tailed Grade Distributions (Rivoirard,2012) was used to estimate grades for Au and Ag. Cross variograms were produced for the truncated variable Au<6 g/t and the indicator for Au>6 g/t. Cross variograms were produced for the truncated variable Ag<110 g/t and the indicator for Ag>110 g/t. The variograms for the truncated variables were marked by low to moderate nuggets and longer ranges than might be achieved for the variogram of the untruncated variable. This is because the truncated variable has removed the noise / pure nugget effect associated with the high / extreme grades. The indicator variogram accounts for that part of the grade distribution associated with high and extreme grades.

Top-cuts were applied to Cu, Zn, Pb and S by reviewing histograms and probability plots.

Grade was estimated into a 3 m x 3 m x 3 m volume block model using Ordinary Kriging for Cu, Zn, Pb and S and the Top Cut Method for Au and Ag.

Swath plots were reviewed to assess semi-local scale reliability of blocks relative to input data along bench, easting and northing slices. Mean grades of inputs and outputs were compared and were within 2% of grades globally for Au, Ag and Cu. For Zn, Pb and S, there were larger differences with blocks reporting grades between 13% and 19% lower than input data. Histograms and probability of inputs and outputs were compared to assess level of smoothing. Visual validation of cross sections showed that blocks reflect the grade tenor of input data.

The estimation methodology used for Au and Ag in the vein model is believed to be appropriate because it reduces the influence of extreme values, while retaining the metal associated with them, and distributing it to the most appropriate blocks.

The MRE for the Shahumyan Deposit has been classified as Indicated and Inferred Resources based on the guidelines specified in the NI 43-101.

The MRE has been compared with that previously reported for the resource by CSA in July 2013 in “NI 43-101 Technical Report Shahumyan Project, Kapan, Republic of Armenia” dated 29th July 2013 and filed on SEDAR. The comparison is as follows:

- All mineralisation wireframes were created to honour a minimum true width of 1.8 m

- A different AuEq equation was used to interpret the mineralisation and estimate grades.

- Underground channel sampling practises were verified and validated onsite and data from them were included in the current MRE (referred to as UC samples). UC samples were excluded from the previous MRE due to concerns over sample quality at the time of the site visit.

- A regression was used to estimate density from a well-defined relationship with AuEq grade, rather a single density average for all resources within the vein model.

- Mining depletion between 1 February 2013 and 31 December 2013.

- The current MRE is based entirely on modelled veins (wireframes). Approximately 34% of the tonnage and 28% of the Gold ounces from July 2013 MRE was estimated using probability methods.

The updated MRE has a 7% decrease in tonnes and a 3% increase in AuEq comprising 24% increase in Au grade, 4% increase in Ag grade, 7% increase in Cu grade and 26% decrease in Zn grade.

Of this, there was an 8% increase in Indicated tonnes, with a 3% increase in AuEq comprising 23% increase in Au grade, 16% decrease in Zn grade; 9% increase in Cu grade, and no change in Ag grade.

There was an 11% decrease in Inferred tonnes, with a 3% increase in AuEq comprising 23% increase in Au grade, 6% increase in Ag grade, 7% increase in Cu grade, and 29% decrease in Cu grade.

1.16.10 Process Plant

The crushing and milling circuits were installed and operated from the early 1970's, when the material being processed was from the previous Centralni mine. The process plant has been upgraded over the past few years by the installation of more modern and larger capacity flotation cells (2005 and 2013) as the feed material became 100% from the Shahumyan deposit. This circuit produces two saleable products – a copper/gold, and a zinc concentrate which are sold to smelters outside Armenia.

1.16.11 Mine Development

Conclusions were drawn from observations made during the site visits by Julian Bennett and include:

- All mining equipment is in a good operational condition.
- Dilution at Shahumyan is currently impacting recoverable grade. There are a number of options to reduce mining dilution, including the use of a new drill rig.

Mine management are fully aware of the need to improve this aspect of the operation.

- A new Simba drill has recently been delivered. This is currently in use and it is expected that it will assist the improvement of stope blasting which in turn should lead to better dilution factors.
- At the time of the visit, capital development was beginning to be a constraint to the opening up of new production areas. To overcome this, a second Boomer rig has recently been introduced underground which has the capacity to complete around 150 m of underground development per month. There are skills issues that are being addressed by increasing the level of training, so that the full utility of this new machine can be realised.

1.16.12 Mine Stability

Conclusions were drawn from observations made during the site visits by Julian Bennett and include:

- No particular issues were noted during the site visit regarding mine stability and ground conditions underground were generally good.
- Some meshing is installed at major tunnel junctions. In the stopes, cable bolting is used to control the hanging wall of the stope in an effort to minimise dilution. It appears partially effective.
- The stopes are filled with mine development waste after completion. This is more an effort to get rid of the waste material underground rather than to provide mine support, although this latter is a secondary result and can only add to the general stability of the area.
- Ambient conditions underground were good, even where diesel equipment was working. Underground air control is with ventilation doors and booster fans.
- Compressors are kept in excellent condition and have reportedly performed well over the years.

1.16.13 Infrastructure

Conclusions as regard infrastructure include:

- Power is stated not to be a constraint at current mine production rates, and is not considered an issue.
- There is no evidence of water shortages.
- Road access is good however there is no direct connection to the national railroad system.

1.16.14 Capital Costs

The capital costs used for estimating the future cash flow model are based on local experience, and actual equipment quotations. CSA considers the capex estimates to be reasonable and acceptable.

1.16.15 Operating Costs

The operating costs used for estimating the future cash flow model are based on current actual costs and factored for the future production rate. A number of operational changes are currently being introduced by management which will impact the operating costs. These changes are mainly intended to optimise stoping widths. Nevertheless, CSA considers the opex estimates to be reasonable and achievable over time.

1.16.16 Financial analysis

The financial analysis has been undertaken by DPM. CSA has reviewed this analysis and believes it to be achievable and fair. Using a discount factor of 5.0%, the net present value ("NPV") of the project is estimated at USD 141.7 M, as presented in Table 5. The project expansion capital for the LOM, excluding sustaining capital and closure costs, is estimated as USD 30.1 M.

Table 5. Project Results Summary

Project Results Summary (LOM)		
Item	**Unit**	**LOM**
Throughput	Mtpa	0.9
Project Life	Years	9
Gold Price	USD/oz	1,300
Copper	USD/lb	3.00
Silver Price	USD/oz	20
Zinc	USD/lb	1.00
Project Economics		
NPV at a discount rate of 5%	USD M	141.7

Production and Revenue Summary (LOM)[(1)]		
Item	**Unit**	**LOM**
Total quantity of material mined/milled	Mt	7.6
Gold grade	g/t	2.44
Silver grade	g/t	37.60
Copper grade	%	0.33
Zinc grade	%	1.00
Overall metallurgical recoveries		
Gold recovery	%	80.9
Silver recovery	%	82.1
Copper recovery	%	96.0
Zinc recovery	%	94.0
Total Net Revenue	USD M	874.7
Site EBITDA	USD M	417.1
Average annual payable production		
Gold	ozs	54,084
Silver	ozs	824,757
Copper	M lbs	5.8
Zinc	M lbs	15.8
Operating and Capital Cost Summary (LOM)		
Item	**Unit**	**LOM**
Throughput	Mtpa	0.9
mining cash cost per tonne of material	USD/t	19.36
processing cash cost per tonne of material processed	USD/t	16.84
Services cash cost per tonne material	USD/t	6.88
royalty per tonne of material processed	USD/t	8.57
admin cash cost per tonne processed	USD/t	8.74
Total cash cost per tonne of material processed	USD/t	60.40
Cash cost per ounce of gold sold, net of by-product credits[(2)]	USD/oz	336
Capital Costs		
Initial capital	USD M	30.1
Sustaining capital	USD M	165.0
Closure and rehabilitation costs	USD M	12.0
Total LOM capital	**USD M**	**207.2**
Project Economics		
Average Annual EBITDA	USD M	52.1
NPV at a discount rate of 5.0%	**USD M**	**141.7**

(1) Excludes first year from life of mine schedule which is deemed to have occurred in 2014.

(2) *Cash cost of sales per ounce of gold sold, net of by-product credits, represents cost of sales, less depreciation, amortization and other non-cash expenses, plus treatment charges, penalties, transportation and other selling costs, less by-product zinc, copper and silver revenues, including realized gains on copper derivative contracts, divided by the payable gold in concentrate sold.*

1.17 Study Recommendations

The recommendations relevant to this study are:

1.17.1 Geological Model

CSA recommends that results from a re-logging program (which was underway at Shahumyan in 2013) including incorporation of recently found detailed paper drillhole geology logs should be reviewed:

- It is recommended that more detail about veins be logged and that this information be used in the construction of wireframes representing vein mineralisation alongside existing wireframes that represent vein and selvedge mineralisation.
- To identify potential mineralised shear zones which have been observed, but about which there is uncertainty about continuity or structural control.
- To improve confidence regarding continuity of interpreted vein models.

1.17.2 Geological Logging

Regarding current DPMK geological logging, CSA recommends:

- Logging of core to 1 m intervals should be replaced with logging honouring geological boundaries.
- Logging to have more detail regarding vein, percentage of vein and other salient detail required to adequately model that mineralisation associated with vein.
- Lithological logging should be better placed to indicate mineralisation and confirm and support continuity of grades between drill holes, which is currently informed predominantly by grade intercepts.
- Training should be provided to on-site geologists to assist identification of mineralisation outside known vein sets as well as increase the amount of detail collected through logging to assist with geological modelling.
- A review of logging should be completed to ensure that all relevant data is recorded in the database that gets used in the interpretation of mineralisation and the MRE.
- Geologists who undertake geological logging should constantly review logging against assay results as field verification and training.
- Efforts should be made to include vein number as part of the logging process.

- Include geotechnical logging as part of the process to provide additional data for mine stability analysis.

1.17.3 Assay QA/QC

No fatal flaws were noted in either the database review or the QA/QC review. Recommendations include:

- Review of field duplicates should continue in order to determine the correlation between original and duplicate results. A bias was noted in the gold field duplicates, which requires ongoing monitoring (only a small sample population was analysed).
- Currently, QA/QC is monitored on a batch by batch basis. QA/QC should be reviewed over longer periods to assist with noting trends in the results (such as analytical drift).
- Umpire 'third party' assaying at an accredited laboratory is recommended as part of a comprehensive QA/QC procedure. QA/QC material should be included with the umpire samples.

1.17.4 Bulk Density

CSA recommends a detailed study to understand the geological controls on density. At present there is little correlation between density and lithology which should be reviewed.

This study could include a program of channel sampling across clearly defined vein exposures and across entire stopes; this would be beneficial in understanding the impact of small density variations across a mined width. Incorporating this with photos, mapping and detailed lithological coding would be beneficial with the objective of verifying the regression used and/or having sufficient data within all major vein groups to rely on flagging of density data. Bulks samples if feasible may also be an appropriate option.

1.17.5 Mineral Resource Estimation

CSA recommends that a review of wireframes be completed to ensure modelled thicknesses using the new wireframe process are honouring the minimum mining width as a minimum.

Narrow vein deposits are often better estimated using 2D methods (grade, thickness). However, there are too many veins at Shahumyan for this to be a practical solution. It is recommended that the current 3D model be compared against a 2D estimate of one of the major veins, in order to test the performance of the current model and to help to refine parameters in future resource estimates.

Investigation of the understating of zinc grades is recommended prior to the next MRE update. This should include, but not be limited to, a review of top cutting to ensure top cuts imposed are not too harsh.

Results from the grade control and reconciliation processes currently underway should be fed back into the MRE to assist with continual improvement.

1.17.6 Grade Control and Reconciliation

Reconciliation of the resource model and grade control models against mined is strongly recommended to improve the confidence in the model. CSA is currently assisting DPMK with a program to develop a grade control model and implement important reconciliation procedures. The results should be reviewed on an ongoing basis, and results integrated into the next resource update as necessary.

1.17.7 Operations 2014

1.17.7.1 Near-mine Development

Near-mine exploration work planned for 2014 is primarily focused on completion of the 2013 near-mine drilling program. Once results have been interpreted, future drilling may occur later in 2014 which will target encouraging intersections and potential extensions to the current mineralised body.

1.17.7.2 Regional Exploration

Regional exploration soil sampling field programs are expected to commence in March 2014, focusing on covering the remaining area within the exposed Middle Jurassic sequence that was not covered during 2013, as well as infill and detailed sampling programs around anomalies identified from the 2013 soil sampling campaign, focusing on the Centralni South and Antarashat South areas. Additional plans for 2014 are:

- Detailed geological mapping of soil anomalies identified during the 2013 soils campaign
- Trenching of soil anomalies
- Based on soil and trenching results; tentative drill testing of encouraging prospects is planned for Q3 2014.

1.17.8 Mining

The mine has undergone a series of upgrades, including the purchase of new mobile equipment. CSA recommends that priority be given to the training of personnel to facilitate these upgrades.

The high level of ROM dilution is recognised by DPM management. This is mainly a factor of the narrow stoping widths. New stope drilling equipment was introduced in 2013, and a programme of further changes to drilling equipment and methods is ongoing. There are varying outcomes as a result of these changes, and the preferred methodology has still not been finalised. Drilling methods alone will not fully address the dilution factor. CSA recommends that a complete review of applicable mining methods is undertaken in order to reduce the quantity of mine waste in the ROM stream.

During a visit in early 2014, CSA noted an improvement in the quality of management since the initial visits in 2013. It should be DPM's aim to continue this trend.

1.17.9 Process Plant

There is no crushed ROM storage between the three crushing stages, and little final crushed ROM storage before the milling circuit. Although this is stated not to cause any hold-ups, some work is advised to improve this area of operation. Management at the process plant are evidently aware of the need for improvements in operations, and given sufficient capital, it is reasonable to expect that improvements in the mill will continue to make headway.

2 Introduction

2.1 Terms of Reference - CSA Global (UK)

CSA Global is an internationally recognised, independent geological and mining consultancy with offices in Australia, UK, Russia, Canada, Indonesia and South Africa. CSA Global (UK) Ltd ("CSA") was requested by Dundee Precious Metals Inc. ("DPM") to prepare an independent Mineral Resource Estimate ("MRE") update for its underground gold and polymetallic Shahumyan mine, located at Kapan in Armenia and operated by a wholly owned subsidiary, Dundee Precious Metals Kapan ("DPMK").

In addition, the update of Mineral Resources for the Project was to be included in a Preliminary Economic Assessment for the Project. Both the reporting of the Mineral Resource update and the preparation of the Preliminary Economic Assessment are the purpose of this Technical Report.

This report is prepared in accordance with the disclosure and reporting requirements set forth in the Toronto Stock Exchange Manual, National Instrument 43-101 - Standards of Disclosure for Mineral Projects ("NI 43-101"), Companion Policy 43-101CP to NI 43-101, and Form 43-101F1 of NI 43-101.

CSA (including its directors and employees) does not have nor hold:

- any vested interests in any concessions held by DPMK or DPM
- any rights to subscribe to any interests in any of the concessions held by DPMK or DPM either now or in the future
- any vested interests either in any concessions held by DPMK or DPM, or any adjacent concessions
- any right to subscribe to any interests or concessions adjacent to those held by DPMK or DPM either now or in the future.

CSA's only financial interest is the right to charge professional fees at normal commercial rates, plus normal overhead costs, for work carried out in connection with the investigations reported here. Payment of professional fees is not dependent either on project success or project financing.

2.2 Principal Sources of Information

See Reference List in Section 27 for a list of documents and reports referenced as part of this MRE Study and Technical Report preparation.

Additional datasets provided by DPMK include:

- 3D models of faults, dykes and topography supplied by Ross Overall

- Mine depletion historical and DPMK surveyed volumes as at 31st December 2013
- Database export (dated 5th December 2013) of collar, survey, assay, lithology, metadata and recoveries supplied to CSA supplied by Ross Overall
- Density data supplied by Ross Overall
- Observations and findings from site visits undertaken by Mr Galen White, Mr Malcolm Titley, Mr Simon Meik, Mr Steve Rose and Mr Julian Bennett
- DPMK Internal Exploration and Geological Reports and memos

2.3 Units

All units of measurement used in this report are metric unless otherwise stated, and are contained in the Glossary of this Technical Report.

All grid coordinate reporting to Armenian government departments must be Gauss Kruger – Pulkovo 1942 Zone 8 projection (“GK8”). A transform of this projection to WGS84 is used on site and for International Reporting requirements.

2.4 Site Visits

2.4.1 Site Visit - Geology

A site visit was undertaken by Mr Galen White (Director and Principal Geologist, CSA) between February 5th-7th and May 29th- 30th 2013 (CSA,2013[1]) to review drilling activities, underground channel sampling activities, geology, data collection methodologies and procedures and to verify project data.

During the site visits, the following review work was conducted;

- Meetings and discussion with the on-site resource geology team, accompanied by Mr Armen Abrahamyan, providing an overview of the department function and responsibilities
- Review of historic geology/resource reports, procedural documentation and policies
- Review of channel sampling maps and channel sampling data in 3D
- Review of the 3D vein model and drilling data in GEMS software
- Shadowing of the on-site Database Geologist, whilst geological logging and assay data was merged in to the database. Review of acQuire database model
- Cross checking of hardcopy assay certificate data with that held in the database
- Review of the core yard facility, and procedures in places for processing drill core
- Meetings with the Exploration Department and an overview of current near-mine exploration activity

- Visit to the SGS laboratory facility in Kapan, including a tour of all facilities
- Inspection of the current operation areas

A site visit was undertaken by Mr Malcolm Titley (Principal Consultant, CSA) between 19 and 22 August 2013 which included a review of site based work relevant to the MRE.

Additional information, including comments and conclusions from the site visit, are summarised in Section 12.5.

2.4.2 Site Visit – Grade Control

Steve Rose, Principal Mine Geologist with CSA, also visited Kapan between 2 September 2013 and 13 September 2013 at the request of DPMK. The purpose of the visit was to review the grade control systems in place. This work is currently in progress.

Observations by Steve Rose regarding underground sampling procedures supported the use of underground channel sampling data in the MRE, as reported in Section 11.1.3.

Work undertaken by Steve Rose included a Gap Analysis. This identified key differences between perceived and actual behaviour of certain geological mining tasks. In summary; approximately 17% of the mining geology tasks were undertaken to levels below what CSA considers to be industry standard and 30% fall below general industry standard practice. Key differences were most notably within the following areas:

- Geological Mapping and Interpretation
- Production Tracking and Control
- Production Reporting
- Reconciliation

2.4.3 Site Visit - Mining

An initial site visit was undertaken by Mr Julian Bennett (Independent Mining Consultant to CSA) between 25 and 28 June 2013 for the purposes of reviewing current mining activity, practises, equipment and facilities. A second visit was made between January 27 and January 30 2014 to again observe mining practises and review the development of a Preliminary Economic Study for the project.

During the site visits, the following review work was conducted:

- Discussions with the DPMK mining team
- Detailed underground visit to review all operations
- Inspection of the underground mine and surface infrastructure
- Review of mining methods and mining equipment

- Visit to the processing facility, including the TMF
- Review of production data and other production documentation.

2.4.4 Site Visit - Process

The first site visit undertaken by Simon Meik (Corporate Director, Processing for DPM) occurred in August 2006, and visits of 3 to 4 weeks duration continued between then and 2009 on a regular basis as DPM took over the management of the operation. The last visit was made between 11th and 20th January, 2013 for the purposes of reviewing current process plant activity, practices and facilities following the latest plant upgrade.

During each of the site visits, the following review work was conducted:

- Discussions and review of all process activities with the DPMK processing team
- Detailed visits to review all processing operations and associated infrastructure
- Review visits to the tailings management facility (TMF)
- Review of production data and other production documentation
- Discussions and review with the Operations Administration team.

Since the last site visit, regular (usually bi-weekly) phone conversations have been conducted with operations personnel to coordinate all ongoing metallurgical activities. These are supplemented by reviewing daily production data. This ongoing review of processing operations has enabled preparation and development of the relevant sections for the PEA for the project.

2.5 Armenian Mining Law

Armenia's mining law came into force in 2002, was again modified significantly in January 2007 when the National Assembly classified all metallic minerals as strategic. In 2005 a new concession law came into force to supplement the mining code. Then on January 1, 2012 a new Mining Code entered into force and the Law on the Concession and the old Mining Code were revoked.

According to the new Code within a year starting from January 1, 2012 all the mining operators were obliged to apply to the Ministry of Energy and Natural Resources for re-issuing the mining right (license, contracts, and act on mining site allotment). With the introduction of the new legislation mining licenses were changed into permits. All the other components of the mining right remained the same.

With the new Code mining rights are provided for two purposes:

- For exploration purposes;
- For mining Purposes.

Two types of exploration permits can be granted: Geological Exploration and Exploration for the Mining purposes. In the case of Geological exploration an agreement from the State authorities must be obtained which will need to be supplemented by a contract. The term of the agreement and the contract cannot exceed three years. In the case of exploration for mining purposes an Exploration Permit must be obtained supplemented by an agreement. The Permit can be provided for up to three years but can be extended three times with two years term each time. The holder of the Exploration Permit has a preference for acquiring mining right for the allotment on which the holder of the permit collected geological information at its own cost.

Mining rights provided for the mining purposes consist of four main documents: mining permit, agreement, mine allotment area and a permitted mining project. With the new changes mining permit term can be extended up to life of mine but not more than 50 years. It is also required to run a resources estimation exercise every 5 years instead of previously required 10 years. Among other changes the new regulation also requires the mine operators to develop a mine closure plan and submit it for approval to the state authorities. Based on the closure plan regular fees will need to be paid to the special fund governed by the State which will be used for financing closure and post closure activities of the mine.

Royalty

The new Code excludes natural Reserves Depletion Fee as part of mining royalty tax. The calculation of the royalty and payment is done quarterly based on the following formula:

Royalty rate = 4% + 100 x (Profit /Revenue x 8)
Royalty = Gross sales x royalty rate

Operational expenses are all expenses related to production with the exception of:

- Capital expenses
- Amortisation and depreciation calculated for tangible and non-tangible assets
- Financing cost including interests for loans.

The royalties are payable before 30th of the month following the reportable period.

Summary information relating to the DPMK licences is contained in Section 4.4.

2.6 Forward Looking Statements

This Technical Report contains “forward looking information” or “forward looking statements” that involve a number of risks and uncertainties. Forward-looking information and forward-looking statements include, but are not limited to, information or statements with respect to the future prices of gold and other metals, the estimation of Mineral Resources and Reserves, the realisation of mineral estimates, the timing and amount of estimated future production, costs of production, capital expenditures, costs and timing of the development of new mineral deposits, success of exploration activities, permitting time lines, currency fluctuations, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, title disputes or claims, limitations on insurance coverage and timing and possible outcome of pending litigation.

Often, but not always, forward-looking information or statements can be identified by the use of words such as “plans”, “expects”, or “does not expect”, “is expected”, “budget”, “scheduled”, “estimates”, “forecasts”, “intends”, “anticipates”, or “does not anticipate”, or “believes”, or variations of such words and phrases or state that certain actions, events or results “may”, “could”, “would”, “might” or “will” be taken, occur or be achieved.

Forward-looking information or statements are based on the opinions and estimates of contributors to this report as of the date such statements are made, and they involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of DPM to be materially different from any other future results, performance or achievements expressed or implied by the forward-looking information or statements.

Such factors include, among others: the actual results of current exploration activities; actual results of current reclamation activities; conclusions of economic evaluations; changes in project parameters as plans continue to be refined; future prices of gold; possible variations in grade or recovery rates; failure of plant, equipment or processes to operate as anticipated; accidents, labour disputes and other risks of the mining industry; delays in obtaining governmental approvals or financing or in the completion of development or construction activities, fluctuations in metal prices, as well as those risk factors discussed or referred to in this report and in DPM’s annual information form under the heading "Risk Factors" and other documents filed from time to time with the securities regulatory authorities in all provinces and territories of Canada and available at www.sedar.com.

There may be other factors than those identified that could cause actual actions, events or results to differ materially from those described in forward-looking information or statements, and there may be other factors that cause actions, events or results not to be anticipated, estimated or intended. There can be no assurance that forward-looking information or statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such information or statements. Accordingly, readers are cautioned not to place undue reliance on forward-looking information or statements.

3 Reliance on Other Experts

The authors of this Technical Report have reviewed available company documentation relating to the project and other public and private information as listed in the “References” section at the end of this Report. In addition, this information has been augmented by first-hand review and on-site observation and data collection conducted by the authors.

The statements and opinions contained in this report are given in good faith. Validation of the historical data obtained from previous operators (which this mineral resource estimate is partly reliant on) is a process of continual improvement. The conclusions and estimates in this report may change over time depending on these improvements, future exploration results, mineral prices and other relevant market factors.

The Qualified Persons take responsibility for the content of this Technical Report and believe it is accurate and complete in all material aspects. However CSA is not responsible for nor has undertaken any due diligence regarding the non-geological/mining technical aspects of this report, which includes;

- Licencing and Tenure information, including;
  - Licence Document 29/183 – 27 November 2012 – Armenian Ministry of Natural Resources
  - Mine Claim Permit 29/183 – 27 November 2012 – Armenian Ministry of Natural Resources
  - Mine Claim Contract (Appendix N1) – 27 November 2012 - Armenian Ministry of Natural Resources

which are translated “signed and sealed” documents relied upon and provided by DPMK, sourced from the Armenian Ministry of Energy and Natural Resources, the details of which are summarised in Section 4.4 of this Technical Report;

- Environmental Study information sourced from;
  - Golder Report 1 - Geghanush Tailings Management Facility Closure Study (Golder Report No. 12514150374.501/B.0)
  - Golder Report 2 - Northern Tailings Management Facility Pre-Feasibility Study (Golder Report No. 12514150374.502/B.0)

Which have been relied upon as presented, and which are summarised in Section 20 of this Technical Report.

4 Property Description and Location

4.1 Background Information

The Republic of Armenia is a landlocked mountainous country of 29,800 km² situated in the South Caucasus between the Black Sea and the Caspian Sea. Located at the juncture of Eastern Europe and Western Asia, it borders Turkey to the west, Georgia to the north, Azerbaijan to the east, and Iran and the Nakhchivan exclave of Azerbaijan to the south.

A former republic of the Soviet Union, Armenia is a multiparty, democratic nation-state with an ancient and historic cultural heritage. Armenia has a population of 2,969,000 (July 2008 est.) which has been declining due to elevated levels of emigration following the break-up of the USSR. This rate of this decline in population numbers has decreased drastically in the recent years due to an influx of returning immigrants.

Ethnic Armenians make up 97.9% of the population. Yezidis make up 1.3%, and Russians 0.5%. Other minorities include Assyrians, Ukrainians, Greeks, Kurds, Georgians, and Belarusians. The predominant religion in Armenia is Christianity (98.7%).

The Armenian economy heavily relies on investment and support from Armenians abroad. Before independence, Armenia's economy was largely industry-based and highly dependent on outside resources. After independence, the importance of agriculture in the economy increased markedly, rising to more than 30% of GDP in the 1990's. At this point more than 40% of total employment was attributed to agro-industries. As the economic situation stabilized and growth resumed, the share of agriculture in GDP dropped to slightly over 20% (2006 data), although agriculture is still an important employer for approximately 40% of the work force.

Armenian mines produce copper, zinc, gold, and lead. The majority of energy is produced from hydroelectric schemes throughout the country.

Educational standards within the country are high with 100% literacy rate reported. The country is served by an extensive network of 7,700 km paved roads, except in the most mountainous districts. There is also a rail network of 840 km.

4.2 Property Location and Accessibility

The Kapan mining area is located in the south eastern corner of Armenia at latitude and longitude 39°12'26"N 46° 24'17"E, 320 km south-east of the capital city of Yerevan. Shahumyan is situated within part of the Tethyan tectonic belt which extends from Southeast Asia to Europe. Figure 1 illustrates the location of the Shahumyan mine.

Figure 1. Location of Kapan Project shown with red circle (Coffey 2009)

4.3 Datum and Projection

All reporting to Armenian government departments must be Gauss Kruger – Pulkovo 1942 Zone 8 projection ("GK8").

In October 2006; DPMK commissioned Spectrum Survey and Mapping Pty Ltd ("Spectrum") in Australia to investigate the coordinate system being used at the operation and to introduce a Real Time Kinematic ("RTK") Digital Global Positioning System ("DGPS") at the project via the existing survey control network. This provided the transformation parameters from Gauss Kruger – Pulkovo 1942 to WGS84.

4.4 Mineral Rights and Tenement Description

4.4.1 Summary

The DPMK held ground in the Kapan area comprises one Mining Licence and one Exploration Licence. The Kapan Exploration License covers 90.6 km^2 with some 12.5 km^2 excised to cover existing populated areas, mine concessions, and related infrastructure as illustrated in Figure 2.

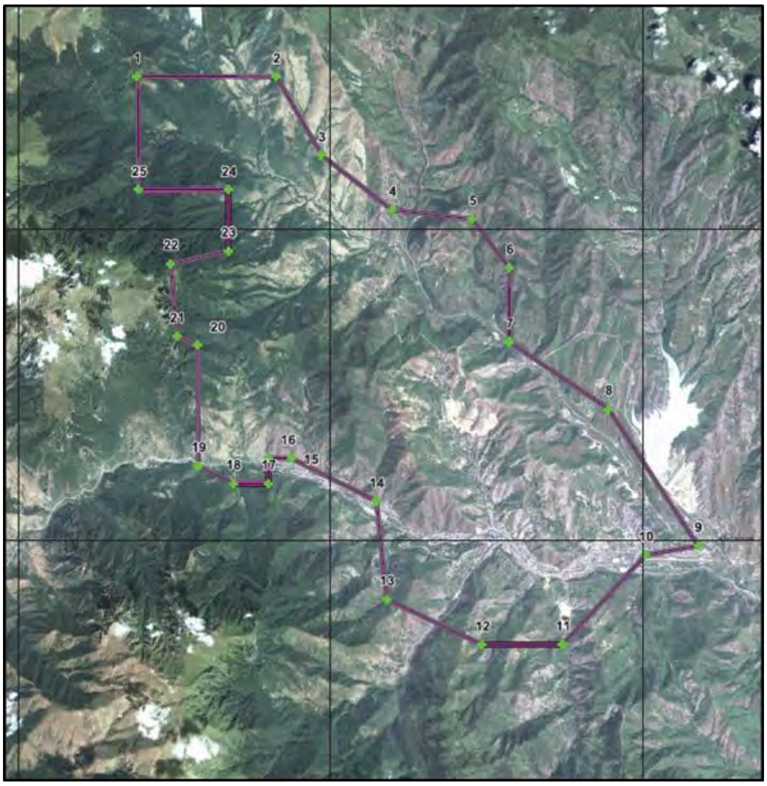


Figure 2. Location Map for the Kapan project licences (DPM,2013).

The current 90.7 km^2 Kapan Exploration License extends from the town of Kapan along a northwest trending valley. Two Mining licences are encapsulated within the Exploration Licence and consist of the Shahumyan Mine and Shahumyan Mill Licence of 2.555 km^2 (which covers the area of the Shahumyan Mining operation and the immediate surroundings). Application with all required documentation for extending the term of the license till April 2016 is submitted and is waiting for approval by the Ministry of Energy and Natural Resources as per the current mining code.

DPMK has operated the Shahumyan Mine concessions under a special mining licence in effect since 2006. Within this mining licence is a licence contract which gave DPMK the right to exploit the Shahumyan Mine Concession until 2011 and the right to exploit the Centralni mine concession until 2017. In 2008 the Centralni Mine license was relinquished due to depleted mineable reserves. In an agreement reached in November 2012, DPMK's Shahumyan mine licence has been extended until 2020.

4.4.2 Mining Permit Terms and Conditions

The following changes to the Mining Permit terms and conditions were negotiated under this new agreement:

- The term of the mine concession will be amended to reflect future evaluated minable reserves.

- Agreement for extension and modification of the license agreement once new projects are approved is to be based on newly approved underground and open cut reserves. Note these reserves are not Mineral Reserves in the context of NI 43-101 but in accordance to local Armenia reporting standards.
- Agreement is required between the Republic of Armenia and DPMK prior to modification of the depletion schedule.
- The current production schedule has been adjusted to 600,000 tonnes mined material production per annum.
- During periods of unusual economic conditions and very low commodity prices the depletion schedule will be automatically adjusted down to reflect mining and milling time lost due to these conditions.

Provision is made under the terms of the Mining agreement between DPMK and the Armenian Government for environmental protection that includes air, water, ground and subsurface environments. EIAs for mine mill and auxiliary facilities were developed and approved by the relevant authorities for continuous operations. Among others, water use and discharge permit, air emissions permit and waste generation and deposition permits are the most significant. All the required environmental licenses and permits are currently attained and updated.

4.4.3 Royalties

There have been several changes in mining legislation in Armenia recently. One of the biggest changes is that the nature utilization payments were replaced by royalties. As a result, the royalty formula has changed.

The royalty is now calculated based on the following formulae:

Royalty rate = 4% + 100 x (Profit /Revenue x 8)

Royalty = Gross sales x royalty rate

Table 6. Tenement corners for the Shahumyan Mining Permit (GK8 grid)

Point ID	Coordinates	
	X	Y
1	8,623,300	4,345,000
2	8,622,650	4,344,500
3	8,623,850	4,342,500
4	8,624,170	4,342,275
5	8,624,250	4,342,275
6	8,624,250	4,342,500
7	8,624,350	4,343,000
8	8,624,100	4,343,800
9	8,624,575	4,345,130
10	8,624,575	4,343,925
11	8,624,025	4,348,975
12	8,624,000	4,344,425
13	8,624,350	4,344,425
14	8,624,350	4,344,500
15	8,623,575	4,345,130

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Exploration and Mining Licences are located in the Syunik State in south eastern part of the Republic of Armenia. The Exploration Licence is centred on the town of Kapan which has about 40,000 inhabitants. The two Mining Licences are located to the north (Centralni) and north east (Shahumyan) of the town and the Kapan Mine mill and associated infrastructure is adjacent to and east of the town. There are numerous villages in the Kapan area which can be accessed by asphalted road. The town of Kapan itself is situated 320 km from the capital of Armenia, Yerevan. This asphalt road is the single main N-S transport route used for freight in and out of Armenia. There is an airport with a paved runway at Kapan however it has not been functioning since the Armenian Azeri war (ceasefire 1994).

The region represents a typical mountainous area located at the end of Arachadzor mountain range, which is the south-eastern branch of the Bargushatk mountain range. The area is characterized by split relief up to 3200 m, and overall altitude of 700 m to 3200 m ASL. The northern and north-western parts of the Kapan mining camp are situated between the south to south-easterly flowing Rivers Voghchi and Khalaj.

5.2 Infrastructure

Due to the proximity of the Exploration Licence to Kapan, various resources are relatively easy to obtain. Electrification of the Kapan region is provided via connection with the United Energy of Transcaucasia system and provides a relatively uninterrupted supply. In addition, there are numerous hydro-electric power stations in the region. The Kapan town and a number of other inhabited localities are also connected to the natural gas reticulation system of Armenia. Both potable and irrigation water are readily available throughout Kapan for industrial use and drilling purposes.

The current surface rights are sufficient for a mining operation. The availability and sources of power and water are summarised in Sections 18.1 and 18.2 respectively. Mining personnel are sourced locally and there are non-Armenian Nationals employed at the operation. Tailings storage areas, waste disposal information and processing information are contained in Sections 20.3, 20.1 and 17.1 respectively.

5.3 Climate

The climate of the region is continental with a long lasting warm to hot summer, which is often dry, and short mild winters often with moderate amounts of snow. Meteorological data from Kapan Met Station is available from 1961 to 1990 and indicates an annual average precipitation of 590 mm with the wettest months from March to June. January is the coldest month with an average minimum of -4.3 ^{o}C and an average maximum of 7.7 ^{o}C. August is the warmest month with an average minimum of 17.3 ^{o}C and an average maximum of 29.6 ^{o}C. The field season is generally active between March and November and at other times much of the concession is snow covered.

6 History

6.1 Pre-DPMK Exploration History

The ownership and production history of the Shahumyan Deposit and neighbouring Centralni Deposit may be summarised as follows:

Artisanal mining of copper mineralisation can be dated in the Kapan area at the Centralni Deposit as far back as 1843 and in the Shahumyan area, to 1863. Numerous small-scale mining operations worked copper showings up until the early 1900's.

In 1905 the French controlled "Cupro Metal" began mining operations at the Centralni area. Operations continued until the early 1920's. At this time initial exploration for polymetallic mineralisation commenced within the Shahumyan area.

Industrial scale exploitation of the Shahumyan deposit by the state owned "Shahumyan Mine Group" was recorded from 1923 up until the early 1950's. Mining was conducted on 4 veins in conjunction with exploration drifts on showings in the north and west of the Shahumyan deposit. An estimated 230,000 tonnes with a head grade of approximately 1.9 g/t Au, 40 g/t Ag, 1.0% Cu, 0.4% Pb and 4.5% Zn was extracted.

From 1953 onward, mining was focused on the neighbouring Centralni Deposit and the Shahumyan Deposit was transferred into an exploration phase. After discovering an extension of the mineralised body to the north of existing mining operations in the Shahumyan Mine in 1962, the development of the 700, 780, 820 and 860 exploration adit levels commenced.

The first initial appraisal of the Shahumyan deposit came in 1976 with the first GKZ mineral resource estimation. This came about as a result of a large, focused exploration program that comprised of underground exploration via shafts and drifts in combination with underground and inclined surface drilling over a 15 year period.

Development of the Shahumyan mineralised body on levels 860 and development of north and south exploration shafts which gave access to 600, 500 and 400 levels, continued until work ceased around 1990 due to the collapse of the former Soviet Union. Between 1962 and 1991 approximately 300 km of exploration drilling and 35 km of mine development within mineralisation was completed. Exploitation of the Centralni Deposit between 1953 and 1991 produced approximately 27.8 Mt of mineralisation at a head grade of 1.23% Cu.

After a hiatus in works, the Armenian privatised entity "Kapan Ore Mining and Processing Enterprise" reopened the Shahumyan mine and began further exploration of the lower levels in 1994. In 1995 a decision was made to start production in the southern section of the mine and prepare the process plant for polymetallic mineralisation. Shahumyan mineralisation was processed in batches whilst the main mineralised feed remained from the Centralni Deposit.

From the Centralni deposit, between 1991 and December 2003 approximately 2,165 Kt of material was mined from underground at 0.81% Cu and between 2000 and December 2003 an estimated 329 Kt mined from an open cut at an average grade of 0.44% Cu.

Kapan Ore Mining and Processing Enterprise declared bankruptcy in 2004. The bankrupted State Centralni Mine was purchased by Vatrin in June 2004 who continued mine production at Shahumyan and Centralni until a formal take over by Deno Gold Mining Company on 1 Jan 2005.

6.2 DPMK Ownership History

DPM acquired 80% of Vatrin, which was the 100% owner of Deno Gold Mining Company, in August 2006 and increased its ownership to 95% late in 2007, and to 100% in 2010. From the recommencement of production in 1994 until DPM's initial acquisition approximately 1,600,000 tonnes of material were extracted at a grade of 1.8 g/t Au, 30 g/t Ag, 0.27% Cu, 0.24% Pb and 1.4% Zn.

Centralni Mine production from January 2005 to 2008 was 137 Kt at 0.68% Cu from underground mining. The Centralni mining licence was relinquished in 2008 and the mining rights handed back to the Republic of Armenian Government.

Deno Gold has operated the Shahumyan Mine under a special mining licence in effect since 2004. The Kapan Exploration License (350 km^2) was awarded to Deno Gold in April 2007. In December 2010 DPM completed its acquisition of Vatrin.

Deno Gold changed its name to Dundee Precious Metals Kapan in 2013 (DPMK). DPMK is a wholly owned subsidiary of DPM.

6.3 GKZ – Previous Mineral Resources Estimates (non-CIM complaint)

Historic resource estimates (non CIM compliant) were typically completed every 4-8 years from 1976 onward however most were internal checks and were not fully accepted by Armenian state authorities. The first approved estimate concerned the main economic elements and also reported values for cadmium, indium, tellurium, selenium, gallium and sulphur.

All GKZ accepted mineral resource estimates for the Shahumyan Deposit follow a similar structure, and those completed by Kolchedan in 2004 (Bagdasaryan,2004) presented in Table 8, updated the "Ore Reserves Assessment" authorised by the Armenian State in 1999 (RA, SRC Report No 67, 29 December 1999), to the extent known, are based on a polygonal grade estimation technique and are classified under GKZ accepted terminology and not terminology accepted and set out in Section 1.2 and 1.3 of NI 43-101. A modification to the 2004 estimate was completed by GeoEconomica in 2008 after depletion using as mined contours, presented in Table 9.

Table 7. Historic mineral resource estimate, 1976.

1976 Shahumyan Resource Statement						
Resource class	**Tonnage**	**Au_ppm**	**Ag_ppm**	**Cu_%**	**Zn_%**	**Pb_%**
C1+ C2	5,704,000	3.80	66.41	0.61	3.01	0.20

1976 Shahumyan Resource Statement – Accessory Elements							
Resource class	**Tonnage**	**Cd_ppm**	**In_ppm**	**Te_ppm**	**Se_ppm**	**Ga_ppm**	**S_%**
C1+ C2	5,704,000	266.8	9.7	63.4	13.8	16.4	7.0

The last accepted resource statement prior to DPMK's involvement was in 2004.

Table 8. Last resource statement prior to DPMK's involvement (Kolchedan,2004).

2004 Shahumyan Resource Statement						
Resource class	**Tonnage**	**Au_ppm**	**Ag_ppm**	**Cu_%**	**Zn_%**	**Pb_%**
C_1+ C_2	18,600,000	2.48	49.18	0.62	2.48	0.16

Shahumyan Resource Statement – Accessory Elements							
Resource class	**Tonnage**	**Cd_ppm**	**In_ppm**	**Te_ppm**	**Se_ppm**	**Ga_ppm**	**S_%**
C_1+ C_2	18,600,000	246.6	9.0	34.0	5.1	15.7	7.1

Table 9. Shahumyan Resource statement completed by GeoEconomica, Yerevan, 2008

2008 Shahumyan Resource Statement						
Resource class	**Tonnage**	**Au_ppm**	**Ag_ppm**	**Cu_%**	**Zn_%**	**Pb_%**
C_1+ C_2	17,719,000	2.81	54.25	0.66	2.72	0.16

2008 Shahumyan Resource Statement – Accessory Elements							
Resource class	**Tonnage**	**Cd_ppm**	**In_ppm**	**Te_ppm**	**Se_ppm**	**Ga_ppm**	**S_%**
C_1+ C_2	17,719,000	247.49	8.53	34.17	4.98	14.5	7.1

Currently it is part of DPMK Mining's operating agreement that a resource update is completed and reported to the Armenian State Authorities every 5 years.

6.3.1 GKZ - Estimation Methodology

All GKZ accepted resource estimates for the Shahumyan Deposit follow a similar structure and are based on a polygonal technique and are not CIM compliant. A Qualified Person has not done sufficient work to classify the historic estimates as current mineral resources or mineral reserves and DPMK is not treating these historic estimates as current mineral resources or mineral reserves.

The classification of mineral resources under the GKZ differs to that of NI 43-101.

According to the level of geological knowledge, the GKZ system identifies four levels of resources, in order of decreasing knowledge (A, B, C1, C2) and three categories of 'prognostic' resources (P1, P2, P3). It is important to note that resources of categories A and B are only reported for areas of detailed study and for confirmation of C1 resource estimates.

According to the state of completion of study of the 'modifying factors' in the GKZ classification system, two levels of definition of modifying factors (which are understood in the same sense as modifying factors under NI 43-101) are identified. These correspond to a lower level of detail required for a "Technical-Economic Justification (TEO) of provisional conditions" document (a pre-feasibility study) and a higher level of detail required for a "TEO of permanent conditions" (a full feasibility study). These two documents correspond respectively to what can be considered an “estimated" mineral deposit and a "fully explored" mineral deposit under the GKZ.

Resources are classified into two groups or categories reflecting their economic significance, - "balance" (economically exploitable) and "off-balance" (potentially economic). In this manner, resources of fully explored and estimated deposits are reported as balance resources (economically exploitable) or off-balance resources (marginal or potentially economic) after completion of the State audit process, following which they are also included in the State balance. Of importance to note is that in the GKZ classification system, the quantity and quality of mineral resources are estimated only from geological criteria, and those estimated with consideration of modifying factors (without adjustment for dilution and losses) are both referred to by a single term - **"resources"**, which does’ not comply with NI 43-101.

The approach can be summarised as follows:

- After sectional interpretation, the results of all samples from a particular vein (drill holes and face samples) are collated and plotted on 1:200 scale sample plans.
- Resource contours are defined using a gold equivalent formula. All samples within the contours are defined as being above cut-off however there is allowance to include internal dilution to keep continuity within the estimate.
- Contours are then plotted on a vertical vein projection; separate mining blocks are allocated within these contours. Sub-economic portions of the deposit are also denoted at this point.
- Sample intervals are corrected to true thickness. The average resource block contour thickness is then calculated.
- A cut-off is applied by reducing any samples that exceed the upper 90th percentile back to the highest value within the remaining sample population.
- The surface area, volume and tonnage of a resource block are then estimated.

- Average grades are estimated using sample length as a weighting value.
- Metal resources are then estimated and allocated a resource category based on predefined criteria. Typically C1 reserves are attainable where a resource sits between two sampled mineralised development drives with drill hole intercepts in between. C2 category is the highest possible class possible using only drill hole intercepts.

Due to the nature of the Shahumyan deposit (polymetallic, narrow veined, structurally controlled) it is designated a complexity class of 3. This classification is derived from a study into the sample distribution within the deposit and the morphology of the deposit. This implies the Shahumyan deposit can never be quoted higher than a 'C' class of resource confidence. C1 category resources are defined as having a Confidence of ± 20 – 40% and C2 Confidence ± 40 – 60%.

6.4 Coffey 2009 - Mineral Resource Estimate (filed on SEDAR)

In 2009 Coffey Mining Consultants prepared a Mineral Resource Estimate for the Shahumyan Project detailed in an NI43-101 Technical Report titled "Technical Report for the Kapan Project, Armenia" dated 30 March 2009 and filed on SEDAR.

All data used in the MRE was collected prior to January 2009 and elements estimated were Cu, Au, Ag and Zn.

The MRE was based upon surface drilling, underground drilling and underground sampling. Only 77 of these surface holes were drilled by DPMK, totalling 12,832.55 m. The remaining material was historic. Only the DPMK data had associated QA/QC information, the results of which demonstrated a good level of precision and accuracy in the assay data.

Following statistical comparison of each data type Coffey decided to exclude the channel samples from the estimation dataset, due to the elevated mean grade as compared to other sampling mediums, and the introduction of grade bias.

6.4.1 Coffey – Estimation Methodology

The following methodology is summarised from the report filed on SEDAR:

- A gold equivalent grade ("AuEq") was calculated, using at the time, current USD US metal prices as set out, below:
  - Cu = USD 5,511.6/t
  - Au = USD 850/oz
  - Ag= USD 16/oz
  - Zn = USD 2,204.6/t

  These values were used as the basis for the following equation for AuEq:

  - AuEq = Auok + (Cuok x 2.017) + (Agok x 0.019) + (Znok x 0.807)

- This AuEq grade was used to constrain the model, based upon an AuEq indicator grade of 0.25 g/t or greater. The indicator model was based on a 5 x 5 x 5 m block size. The variogram used to assign the AuEq grades was based upon 5 m down hole composites derived from the historic drilling and channel sampling dataset. Blocks with a 50% probability of >0.25 g/t AuEq were considered mineralised and allocated a zone code of 1.
- Top cuts were applied to all elements.
- A mean bulk density value of 2.60 g/m^3 was applied to all blocks in the block model.
- Correlelogram variograms were used to investigate 3D spatial grade continuity.
- The block model for grade estimation was constructed of 10 x 10 x 10 m cells. Grade was estimated using ordinary kriging ("OK"), with inverse distance weighting ("IDW") and nearest neighbour ("NN") estimates run for comparison and validation purposes.
- A distance restriction was applied to high grade data, only allowing it to be selected in the first search pass. For subsequent search passes a top cut was applied to this data for all elements.
- Block model validations suggested the estimates to be globally robust, honouring the input date used in the estimations.

6.4.2 Coffey - Mineral Resource Reporting

The mineral resource was classified as Inferred, and was reported cut-offs (presented in Table 10).

It is unclear from the available literature whether the resource was reported at a single grade cut-off, which is a requirement under NI 43-101.

Table 10. Mineral Resource Statement (Coffey 2009)

Cut-off (AuEq g/t)	Tonnage (Mt)	AuEq (g/t)	Cu (%)	Au (g/t)	Ag (g/t)	Zn (%)
0.00	466	0.93	0.09	0.37	6.50	0.32
0.25	407	1.05	0.10	0.42	7.40	0.37
0.50	336	1.19	0.11	0.48	8.39	0.41
0.75	227	1.47	0.13	0.61	10.32	0.49
1.00	147	1.80	0.15	0.79	12.62	0.57
1.25	98	2.14	0.17	0.99	14.99	0.65
1.50	70	2.45	0.18	1.19	17.00	0.72
1.75	49	2.80	0.19	1.43	19.14	0.78
2.00	36	3.13	0.19	1.68	20.87	0.83

NB: AuEq US price assumptions; Cu USD 5,511.6/t, Au USD 850/oz, Ag USD 16/oz and Zn USD 2,204.6/t.

6.4.3 Conclusions and recommendations

Coffey made the following conclusions and recommendations as part of their work:

- Historical and channel data was considered problematic and a high-risk item for the estimation.
- Comparative reporting of the various models within the area of the new DPM drilling indicated a significant increase in grades when channel sampling is included.
- Globally the mineralisation volume was reasonable.
- A robust geological model is required for high confidence grade estimation.
- The variography was poorly structured and considered to be of low confidence.

Coffey Mining provided the following recommendations:

- Specialized structural geology mapping carried out by Jigsaw Geoscience should be the basis of a thorough review of the current geological model.
- Wireframing of domains is recommended.
- Wireframing of significant high grade “vein” should be treated as a sub-domain.
- Based on production plans provided by DPM, the likely depletion is insignificant versus the total model (estimated at <0.4%).
- An updated resource estimation should be completed for the Vein 5 area using the DPM only data set.

7 Geological Setting and Mineralisation

7.1 Regional Geology

The Kapan project is located in southern Armenia within the relatively under-explored Lesser Caucasus. This mountain range is part of the extensive Tethyan orogenic belt and is located within a paleo-Tethys suture zone. The Tethyan Orogeny began approximately 150 Ma as Cimmeria was subducted beneath Laurasia closing the Tethys Sea. The Tethyan orogenic belt can be traced from Middle Europe, across the Middle East to the Indian Ocean, as illustrated in Figure 3.

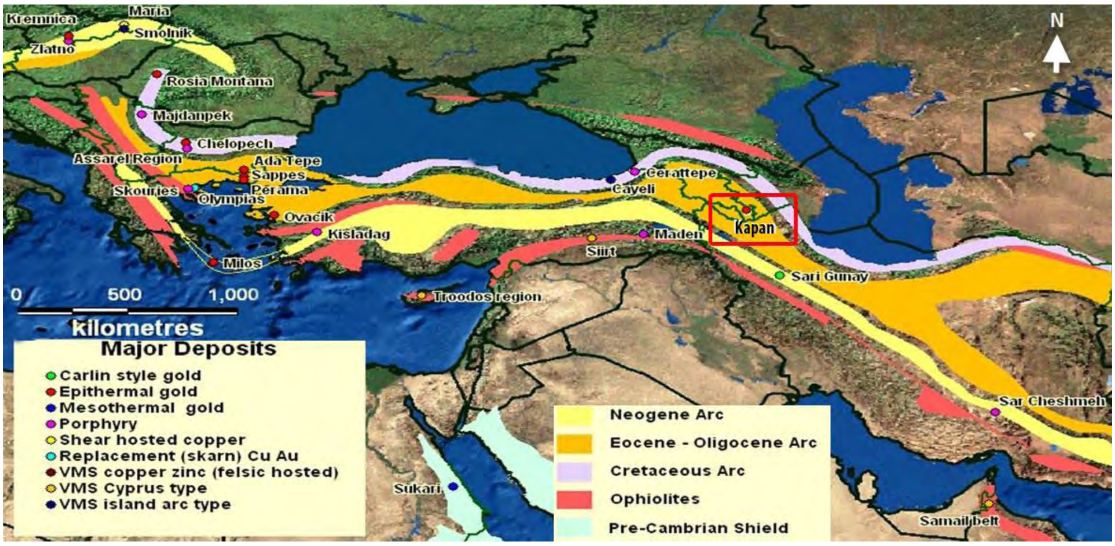


Figure 3. The Tethyan Metallogenic Belt (source Gentor Resources,2013). Kapan project location is outlined in red, central right in image.

Several metallogenic events have been distinguished in the Lesser Caucasus and range in age from Middle Jurassic to Neogene. Upper Jurassic and Lower Cretaceous volcanic and sedimentary formations are localized only to the north-eastern and northern limbs of the Kapan Anticline. Dundee's exploration and mine tenements are situated within the Kapan Anticline.

7.2 Property Geology

The Shahumyan mineralised field is thought to represent a large hydrothermal sulphidation system that was emplaced in connection with the intrusion of an underlying felsic intrusive body. It is thought that Shahumyan sits on the roof of such an intrusive body, large district scale faults allowed hydrothermal fluids to be trapped in certain structurally controlled areas (Tate,2012).

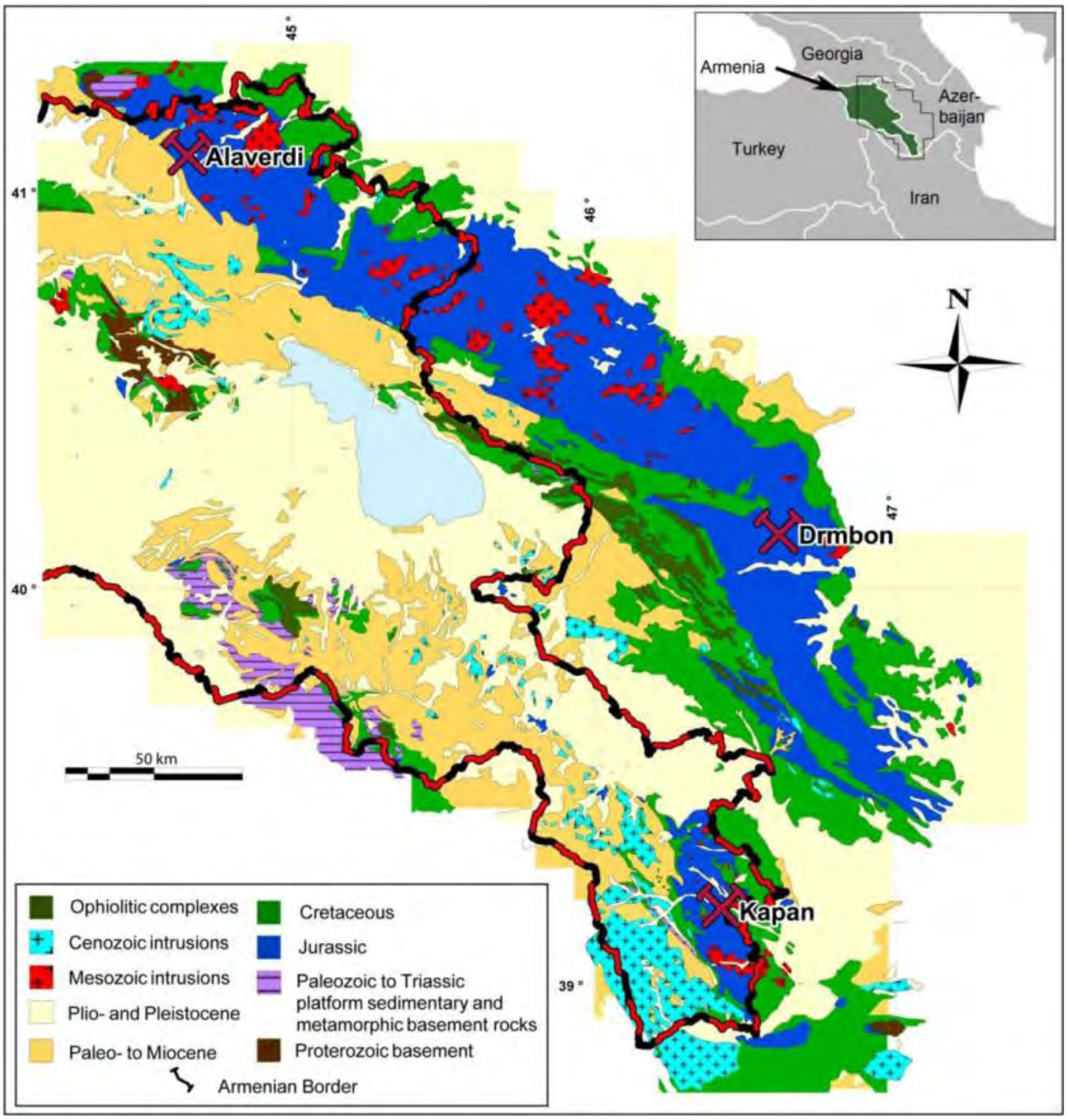


Figure 4. Simplified geologic map of the Lesser Caucasus region in Armenia. The Alaverdi, Drmbon, and Shahumyan deposits are hosted by Jurassic volcanic and volcanosedimentary rocks of the NW-SE–striking paleo-island arc (shown in blue and green) (Mederer et al,2012[1]).

The middle Jurassic host rocks comprises of a suite of intermediate tuffs, flows, breccias and sub-volcanic intrusions which generally are dipping at a shallow angle to the east. Primary bedding and contacts are very hard to identify within the region. Contacts are often faulted, altered and/or mineralised making confirmable contact difficult to prove.

7.3 Local Geology

The Shahumyan deposit exhibits numerous syn-volcanic dykes and sills of andesitic to gabbroic composition. Such intrusions are pre-mineralisation. Pre-mineralisation dykes often have mineralised contacts and are always subject to propylitic alteration or higher grade (typically within contacts of veins or major structures).

Post-mineralisation dykes are gabbroic composition (typically of diabase) and crosscut mineralised veins obliquely. Such dykes possess a NW-SE strike, dip steeply and are 1-10 m in thickness. Post mineralisation dykes are barren nature and have a strong magnetic response.

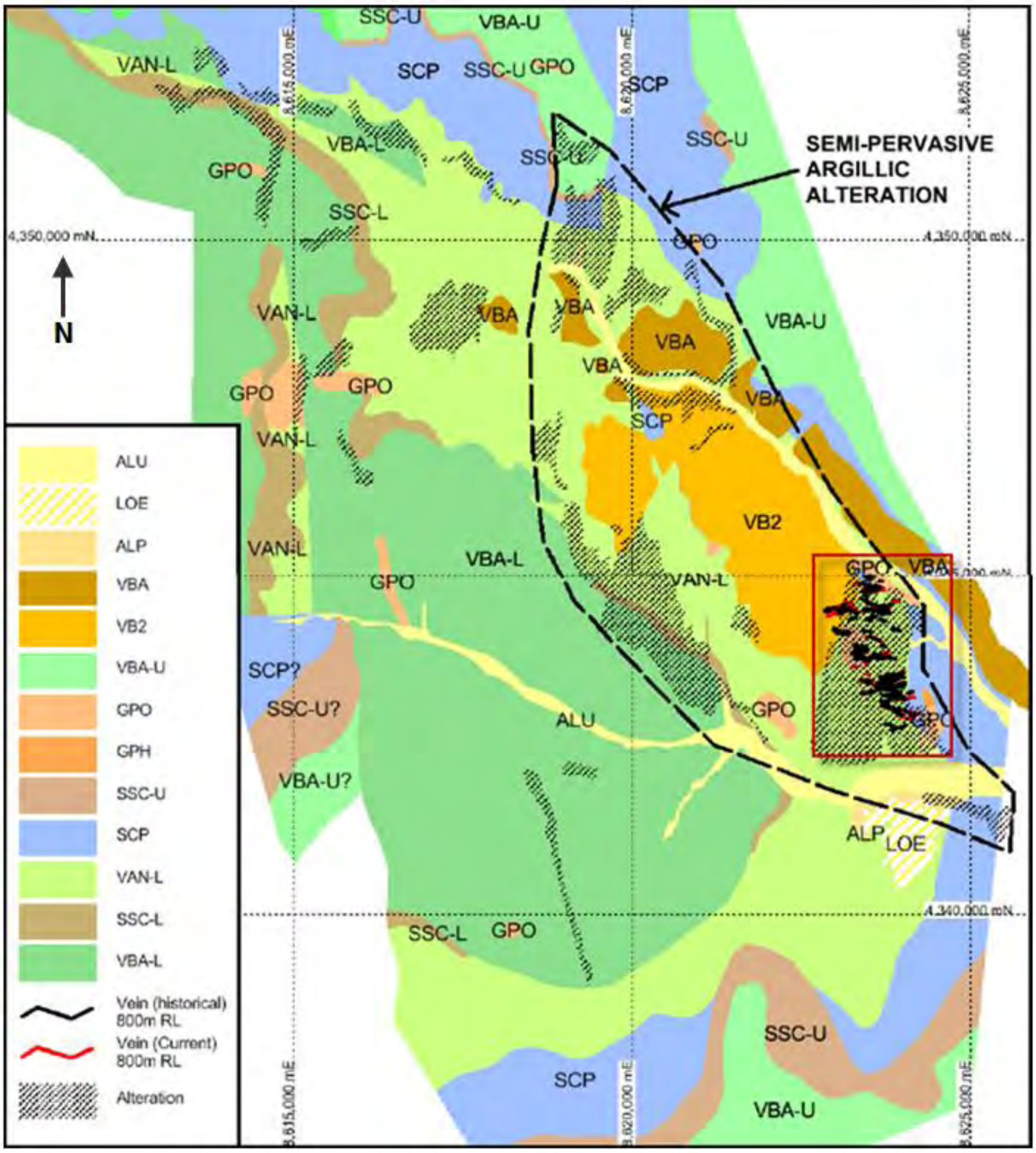


Figure 5. Local Geology at Shahumyan. Soviet alteration mapping overlaid on geological interpretation (Tate,2012). Shahumyan E-W vein set shown and is enclosed in a red box image right.

The middle Jurassic suite is cut by a series of major controlling faults which cut the deposit in a NNW-SSE orientation. Faults are characterised by 2-10 m wide zones that are comprised of fault gauge and mylonitic clay. Faults generally exhibit multiple senses of movement but a clear dextral movement sense can be noted in most.

At Shahumyan mineralisation is hosted by a suite of sub-parallel veins, the current interpretation of the deposit includes in excess of a hundred veins. In many cases the continuity of these veins is yet to be established and future drilling will be important to validate the current model. Regardless, development along and across the vein system has confirmed the prevalence of approximate east-west strikes and steep dips of the veins.

Hydrothermal-altered breccia units are also notable within the deposit makeup. Such breccias are generally found on the eastern flank of the deposit and contain polymictic clasts (andesite, basalt, dacite) within a sulphide rich matrix. Mineralised clasts can be found which are thought to have occurred due to precipitation of sulphides within vesicular lithologies.

Geological understanding of the mineralisation-alteration system is being developed in conjunction with current geological modelling which supports the resource work.

Additionally previous mapping campaigns by historical Soviet operators and more recently by external contractors appear to have conflicting models (Tate,2012). Dundee is endeavouring to establish a robust geological model.

A first-pass paragenetic sequence compiled by Dr Brett Davis (2006) is presented below in Table 11.

Table 11. A mineralogical and structural paragenesis table (modified from Davis 2006).

Mylonite development, East-over-west (?) thrusting of sedimentary cover sequences over volcanic host sequences	
Explacement of mineralisation as east-west striking, mineralised sulphide-quartz extension veins	quartz pyrite sphalerite & galena chalco
Accomodation of minor progressive deformation of the quartz veins	
Coarse grained brassy pyrite veins with coarse-grained sphalerite and quartz infill	
Quartz-only extension veins and breccias	
(Mn) carbonate +/-fluorite extension veins	
Pug-filled faults. Pug varies in colour from white to grey	
Dykes	
Extension I, Accomodated by early thrust structures. Local development of a penetrative low-dipping foliation	
Extension II, Marked by steep, variably striking normal faults. N-S strikes are most common?	

7.4 Mineralisation

Mineralisation within the Shahumyan deposit is characterized by narrow veins (0.2 – 2.0 m), steeply dipping and striking approximately east-west that host a polymetallic (Cu-Zn-Pb-Au-Ag) mineral assemblage. Veins are often found in networks and "zones" that may be shadowed by a wide zone of argillic alteration and disseminated mineralisation. Such zones can be as wide as 25 m (for example vein 5) but typically are 10-15 m wide and appear most common in higher elevations within the deposit.

Mineralisation is primarily semi-massive to massive sphalerite, pyrite, chalcopyrite, galena and bornite and occasionally, found with the latter mineral assemblage of tennantite, tetrahedrite, bornite, enargite, chalcocite, covellite, Pb, Au, Ag, Bi tellurides and native gold and silver (recorded during petrographic analysis). Gangue minerals present are typically quartz, calcite, gypsum, anhydrite and rhodochrosite. Veins exhibit a variety of textures which relate to the numerous stages of mineralisation and tectonism that have occurred.

According to Tate (2012) the mineralogy, scale and distribution of veins and alteration are consistent with the upper parts of a large porphyry style hydrothermal system with the Shahumyan vein system representing a basement carbonate zone within the system. The Centralni stockworks represent the lower transitional parts of a high sulphidation epithermal zone within the porphyry hydrothermal system.

Soviet geologist historically are of the opinion that a clear zonation occurs across the mineralised field with a gradual change of high to low temperature minerals: quartz-pyrite-molybdenite formations (Dzorastan), quartz-pyrite with chalcopyrite (Kurtamyak), quartz-pyrite-chalcopyrite (mine 7-10), chalcopyrite (Centralni mines 1-2, 6), polymetallic (Barabatoom mine), polymetallic and gold (Shahumyan Mine). This would suggest that a high temperature metal source existed to the NW of the mineralised field; however this idea has never been proved conclusively.

Mederer et al (2012) note that sulphur isotope compositions of sulphides and sulphates from the Alaverdi and Shahumyan deposits are in agreement with a magmatic source for the mineralisation-forming fluids; a typical volcanogenic massive sulphide (VMS)-only scenario for these deposits can be ruled out. However, as the juxtaposition of different mineralisation styles has only recently been revealed in complex arc or back arc systems, the involvement of seawater during mineralisation-forming processes in some of the deposits cannot be disregarded.

Middle Jurassic to Lower Cretaceous ages of mineralisation were established by crosscutting relationships and by K-Ar dating of altered host rocks in the Alaverdi, Drmbon, and Shahumyan deposits.

Supergene enrichment is negligible within the Shahumyan Deposit, and only minor secondary copper mineralisation can be found around major structural contacts within the deposit architecture.

Figure 6. Vein 11 from 775 Level. Massive sphalerite within disseminated chalcopyrite associated with massive calcite infill.

7.5 Alteration

Propylitic alteration within the Shuahamyan mine site is widespread. This is typically comprised of chlorite, epidote ±carbonate ±quartz and ±sericite. Within areas that exhibit sulphidation disseminated pyrite is common.

This alteration increases in intensity towards the major structures and veins. Phyllic to argillic alteration is most common particularly within stockworks and is characterized by pervasive quartz-pyrite and replacement of feldspar by sericite and clay minerals. Argillic alteration is a good indicator of proximity to mineralisation. Silicification is also notable in areas of quartz networks and stockworks. A 40Ar/39Ar age of 156.14 ± 0.79 Ma was obtained from magmatic-hydrothermal alunite which is associated with advanced argillic alteration in the upper part of the Shahumyan deposit (Mederer et al,2012).

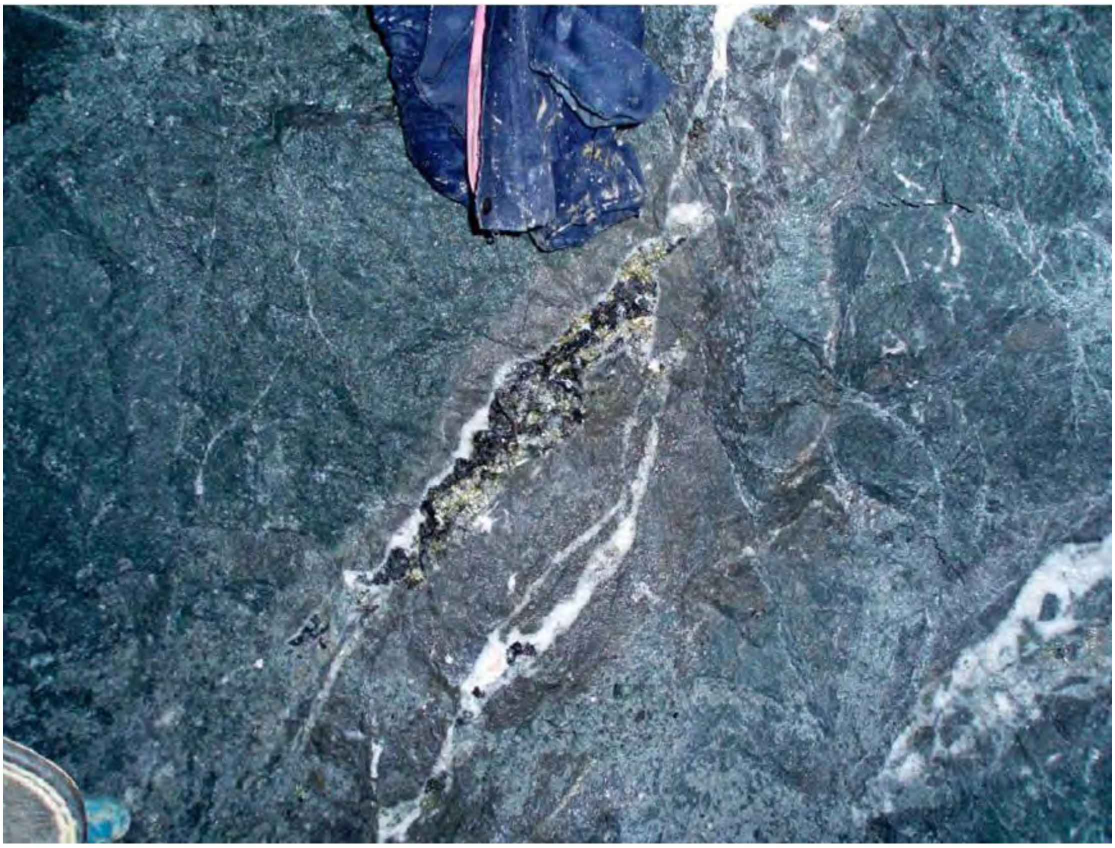


Figure 7. Coarse grained brassy pyrite and sphalerite occurring in association with carbonate with brown alteration halo, V13N, 780 Level (Davis 2006).

7.6 Structure

The regional geologic structure of the Shahumyan field is complicated by numerous regional faults and shears. Evidence of tectonism is widespread and it has occurred continuously throughout the geologic history of the region. Major regional faults include the Metz-Magarin, Khaladzh, Barabatum–Khaladzh, Khotanan, Sayadkar and Bashkend faults. The major tectonic events occurred prior to the Upper-Jurassic period and pre-determined the main structural elements of the field.

The Shahumyan deposit is located along a corridor of Middle Jurassic andesites that sit within a series of north-west/south-east trending faults. Five tectonic blocks can be identified within the deposit, the premier of which within the Shahumyan deposit sits between the Eastern and Centralni Shahumyan faults. Beyond this corridor mineralisation does persist to the east in the form of polymetallic veins and networks which terminates at the upper Jurassic contact. To the west, mineralisation does exist in a few notable areas but again gradually decreases until coming in contact with the Barabatoom-Khaladge fault which marks the contact with the Upper Jurassic. Mineralisation is then found 2 km to the west within the historic Barabatoom mining area. The Shahumyan deposit remains open to the north and south under Tertiary and Quaternary cover.

Veins generally strike east-west; however it is very common for veins to have inflections and junctions with other veins. Veins are generally stacked in an en-echelon fashion dipping to the south at 60-85 degrees.

Overprinting relationships between veins, structures, and alteration suggest that the veins at Shahumyan were emplaced early in the structural history of the deposit. The veins exhibit typical extensional vein textures, including breccia textures, indicating the formation of open space infill by successive mineral species.

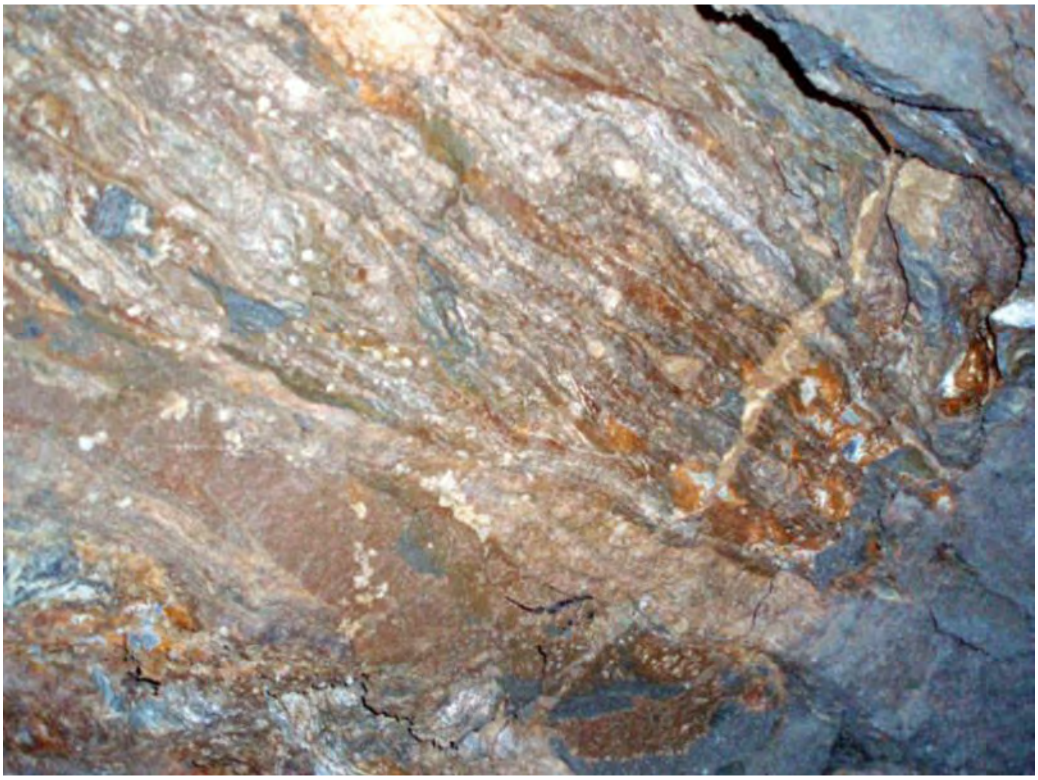


Figure 8. Underground exposure of well-developed mylonitic foliation wrapping porphyroclasts and displaying an apparent sinistral sense of shear (Davis 2006).

Local deformation of the veins is also evident. In some cases the veins are oriented sub-parallel to brittle-ductile shears and are in turn deformed by them suggesting formation of vein-parallel shearing at the time of vein emplacement. These vein-parallel structures are grossly under-represented in current mapping despite their obvious control on void space development visible during mining operations. The majority of veins have an easily discernible shear associated with them. The veins are in nearly ideal orientations for north-south opening under the imposition of an approximately east-west far-field stresses.

Davis (2006) observed at Shahumyan that the Soviet vein interpretation had strongly influenced the local mine model and did not take into account DPMK down-hole orientation data. There appeared to be some discrepancy in the south of the Shahumyan where veins are modelled as dipping northwards as opposed to the more likely scenario of two separate veins dipping south. No geological reconciliation of the veins has been attempted to validate this model. Also low-dipping structures are commonly seen but not represented in the current model.

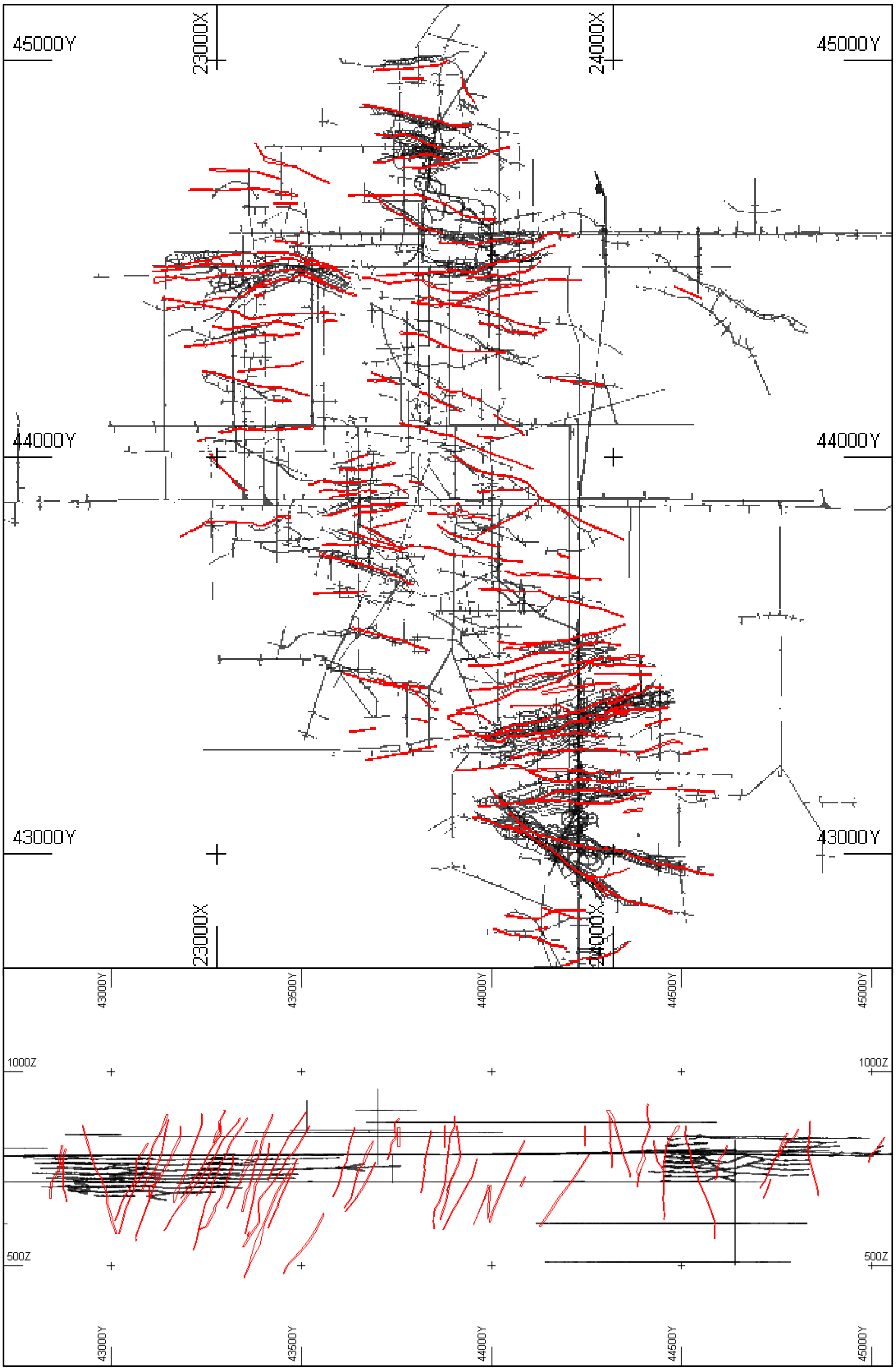


Figure 9. Plan View (top) and Section View at 23,700 mE looking W (bottom) showing Vein Model used in the current MRE (red) and mine developments (black).

8 Deposit Types

In the Kapan mining district, mineralisation dominantly occurs as two styles – as chalcopyrite-pyrite veins and stockworks and poly-metallic vein systems hosted by a volcano-sedimentary rock sequence.

The differences between deposit types are discussed in more detail in Mederer (2012) and are presented from this report in Table 12 below.

Table 12. Key Deposits of the Lesser Caucasus region (Mederer 2012).

Distric	Status Reserves	Host Rocks	Mineralogy	Alteration	Alteration	Ore Body	Age
Kapan District	**Shahumyan** Polymetallic veins		Middle Jurassic subvolcanic quartz-dacite	py, cp, sl, gn, tn, td, tellurides, native Au. Gangue: quartz and carbonates	Phyllic alteration around veins, advanced argillic alteration locally	E-W striking steeply dipping veins	156.14 ± 0.79 Ma (Ar-Ar on magmatic-hydrothermal alunite)
	Centralni East Sulphide stockwork **Centralni West** Sulphide-quartz veins and stockwork	Closed deposits. At least 31 Mt of ore @ 1.16 % Cu mined since 1843	Middle Jurassic volcanic and volcaniclastics Middle Jurassic volcanics and volcaniclastics	cp, col, tn, td, enr, minor lz and gn cp, py, gn, td, tn. Gangue: quartz and carbonates	Argillic alteration and silicification Chlorite-epidote alteration with sericitication in the upper	Stockwork E-W striking steeply dipping veins and stockwork in volcaniclastic rocks	144.7 ± 4.2 (Re-Os on pyrite) 161.78 ± 0.79 (Ar-Ar on hydrothermal muscovite)
Alaverdi District	**Alaverdi** Cu-pyrite veins and lenses	Closed. 5 Mt @ 3.4 % Cu and 0.12 ppm Au	Middle Jurassic tuffs and andesites	cp, py, sl, bn, cc, and subsidiary gn, tn, stn, emp, arg, native Au and Ag, el, apy	Sericite-chlorite and silicification	Subvertical veins in deeper parts,	Uncertain, possibly Upper Jurassic
	Shamlugh Polymetallic veins and lenses	Active open pit mine. 4 Mt @ 3.53 % Cu, 4.96 % Zn, 1.03 ppm Au, 8.1 ppm Ag	Middle Jurassic basaltic andesitic, andesitic and dacitic tuff and lava breccia, overlain by a rhyolite sill (named	cp, py, sl, bn, cc, and subsidiary gn, tn, stn, emp, arg, native Au and Ag, el, mr	Sericite-chlorite and silicification	lithologically controlled ore lenses in upper parts	142 ± 6 Ma and 161 ± 4 Ma (K-Ar whole rock and sericite ages of altered host rock)
	Akhtala Polymetallic veins and lenses	Closed. 1.2 Mt @ 0.58% Cu, 1.67% Pb, 4.48% Zn, 1.3 ppm Au -104 ppm	Middle Jurassic subvolcanic quartz-dacite, andesite and basalt	bar, gn, sl, cp, tn, td, and subsidiary bn, cc, mr, cst, arg, el, native Au and Ag	Sericite-chlorite, pyrophyllite-dickite, silicification	E-W striking subvertical veins, stockwork and stratiform lenses	141 ± 5 Ma and 150 ± 5.5 Ma (K-Ar whole rock age of altered host rock)
Drmbon District	**Drmbon** Cu-pyrite lenses	Active underground and open pit operation. 3 Mt @ 3.8 ppm Au, 0.9 % Cu	Middle Jurassic basaltic andesite to dacite	Py, cp, native Au, hm and subsidiary sl, gn, bn, tn and td	Sericite-chlorite, silicification, hematitization	Lithologically controlled stratiform ore lenses, stockwork underneath	post-Oxfordian (Upper Jurassic)

9 Exploration

9.1 Introduction

Within a 15 km radius of the Kapan exists numerous showings all related to the Centralni (Cu) and Shahumyan (Au-Ag-Cu-Pb-Zn) hydrothermal systems. DPMK is currently systematically evaluating all prospects within the near mine area. Within much of the near mine area the prospective middle-Jurassic unit is overlain uncomfortably by upper Jurassic sediments and due to this prospects may be blind or have limited surface footprints.

Since 2007, DPMK has undertaken regional as well as focused geochemical and geophysical exploration and structural mapping programs. These exploration works are presented in Table 13.

Table 13. Summary of exploration work and baseline studies undertaken by DPMK at the Shahumyan Project, (Deno Gold 2011).

Type of Work	Details	2010			2011				
		Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1
Environmental baseline studies, analysis and assessment	-								
Geochemical regional stream sediments samples assaying	210 Samples								
Geological mapping	200 Samples								
Laboratory analysis of the regional lithogeochemical samples	16,000 Samples								
Detailed lithogeochemical samples assaying	2,310 Samples								
Core samples assaying	19,000 Samples								
Preliminary resources evaluation	-								
Annual information report to the SNCO Geological Fund	-								
Report to the Agency of Mineral Resources	-								

9.2 Satellite Imagery Acquisition and Interpretation

Various satellite image packages have been acquired by DPMK, illustrated in Figure 10. These have been used for work planning to assist in target selection and for developing maps, and include:

- Very high resolution Quickbird satellite imagery is used as a base map for navigation and display purposes.
- Shuttle Radar Terrain Model (SRTM), a moderate resolution radar topography data set that provides excellent regional topography and has been utilized over the project for the interpretation of broad structural trends.

- ASTER technology, which measures reflected spectral data from rock forming minerals (e.g. micas, vegetation water) and assists in establishing alteration haloes and is used to target mineralisation.

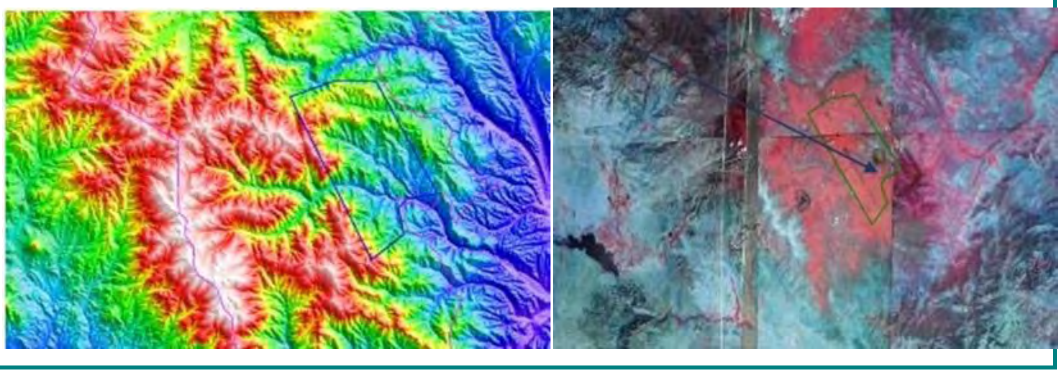


Figure 10. Examples of SRTM (coloured by elevation) (left) and ASTER data (right) for southern Armenia, (Deno Gold,2011). Exploration permit outlines are shown on each image.

9.3 DPMK Airborne Geophysics - 2007

A helicopter borne Versatile Time Domain Electromagnetic ("VTEM") survey was completed by Geotech Ltd of Canada between July and August 2007. The VTEM survey is used to assist in the outlining of conductive features such as massive sulphide mineralisation, wide zones of alteration or graphitic units; and is capable of detection to depths up to 400 m. It is characterized by exceptionally high resolution, precision and accuracy.

A total of 3877 line kilometres at a line-spacing of 80 m were flown across for an approximate surface area of 280 km^2. The processed survey results are presented as 1) total magnetic field and 2) stacked time domain electromagnetic profiles, and include xyz data and survey surface topography, as illustrated in Figure 11 and Figure 12.

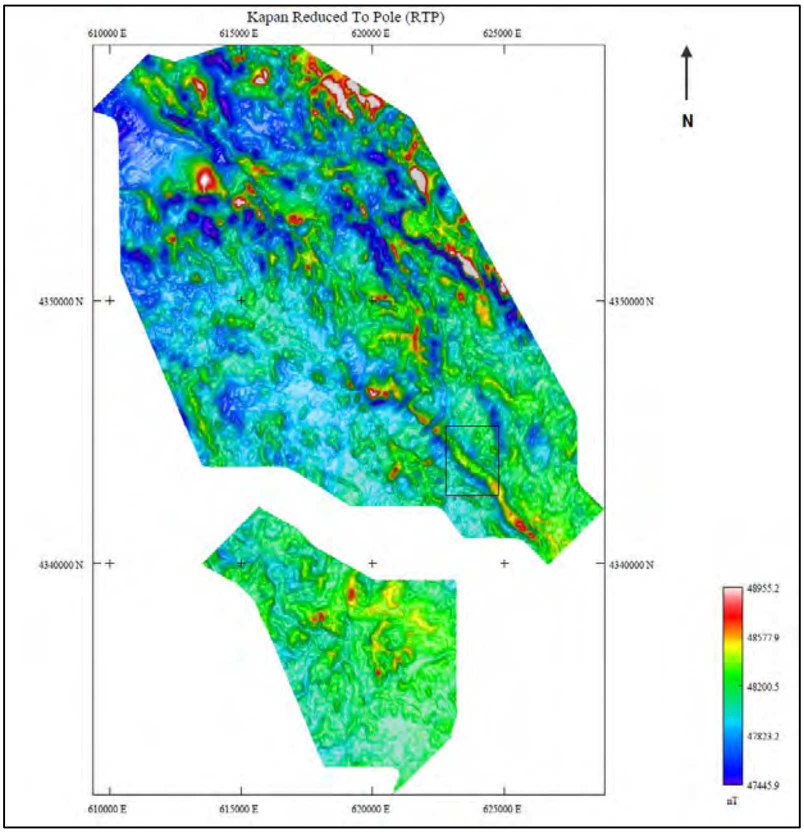


Figure 11. Complete gridded VTEM data: Magnetic data with a Reduced to Pole filter Shahumyan mine location outlined in black, middle right of image (DPM,2007).

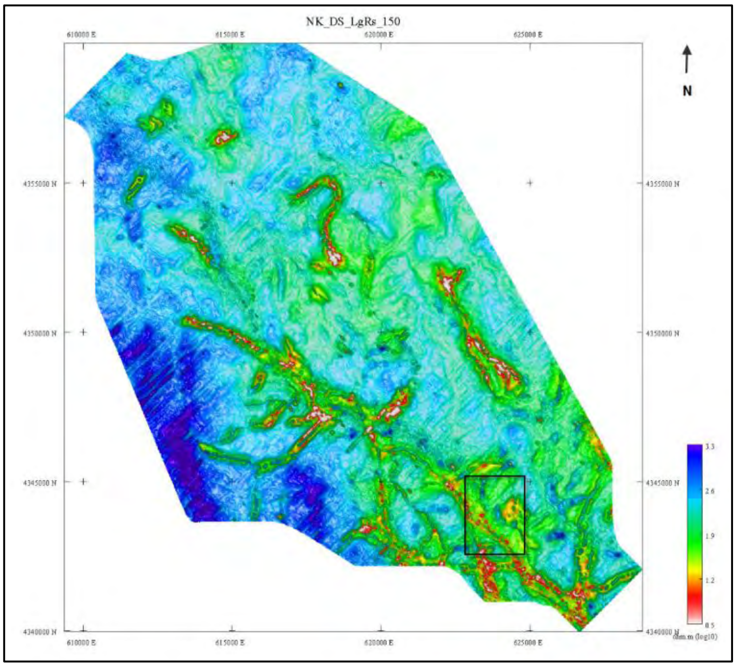


Figure 12. Complete gridded VTEM data: 150 stack time domain EM inversion. Shahumyan mine location outlined in black, lower right of image (DPM,2007).

9.4 DPMK Stream Sediment Sampling

Stream sediment sampling was completed (Deno GMC,2011) from active stream channels, preferably from secondary drainages and only upstream from a primary drainage. Where flowing streams were encountered, sediment trap sites (bars, banks, etc.) were sampled. In dry streams, samples were collected across the tail end of an active sediment bar, parallel to the end of the bar, which ensures that all size fractions were included in the sample. The weight of each sample gathered is approximately 5 kg passing -2 mm sieve fraction. The planned density of sampling was 1 sample per square kilometre, though in high priority areas this was reduced to 1 sample per approximately 3/5 square kilometre. Samples were submitted for BLEG analysis.

The QA/QC sample insertion consisted of a minimum of 20% of the total samples collected and included:

- Replicates: taken proximal to the location of the primary sample
- Duplicates: taken from the same location as the primary sample
- Certified standards: prepared material with certified grades
- Blanks

The soil sampling SOP was supplied to CSA (Soil and Stream Sediment Sampling Protocols 2008.doc) and is of an appropriate standard. Sampling is considered representative. There are no known factors that may have introduced sample bias.

The results of stream BLEG sampling identified Cu, Au and Ag anomalies and potential sources were prioritized for follow-up work.

9.5 DPMK Soil Sampling

Soil sampling was conducted in prospective areas at 50 m intervals along lines oriented north-south with a line spacing of 200 m east-west. In non-prospective areas line spacing was widened to 400 m. A soil sampling orientation program was conducted across known mineralisation boundaries at the Shahumyan Project to determine which size fraction was most suited to exploration at the licence area. It was found that no significant differences in geochemical response existed between the size fractions when viewed as a mineralisation vector, and that the un-sieved samples were as representative as sieved samples. Due to this, samples collected throughout the rest of the campaigns were submitted un-sieved for analysis. The samples were collected from the B-Horizon by digging a pit wherever possible. Samples were submitted for fire assay for gold analysis. Analysis for 51 base metal and pathfinder elements was by two acid digest followed by an ICP-MA/AES finish.

QA/QC procedures were equivalent to the stream sampling program and are summarised in Section 9.4. Sampling is considered representative. There are no known factors that may have introduced sample bias.

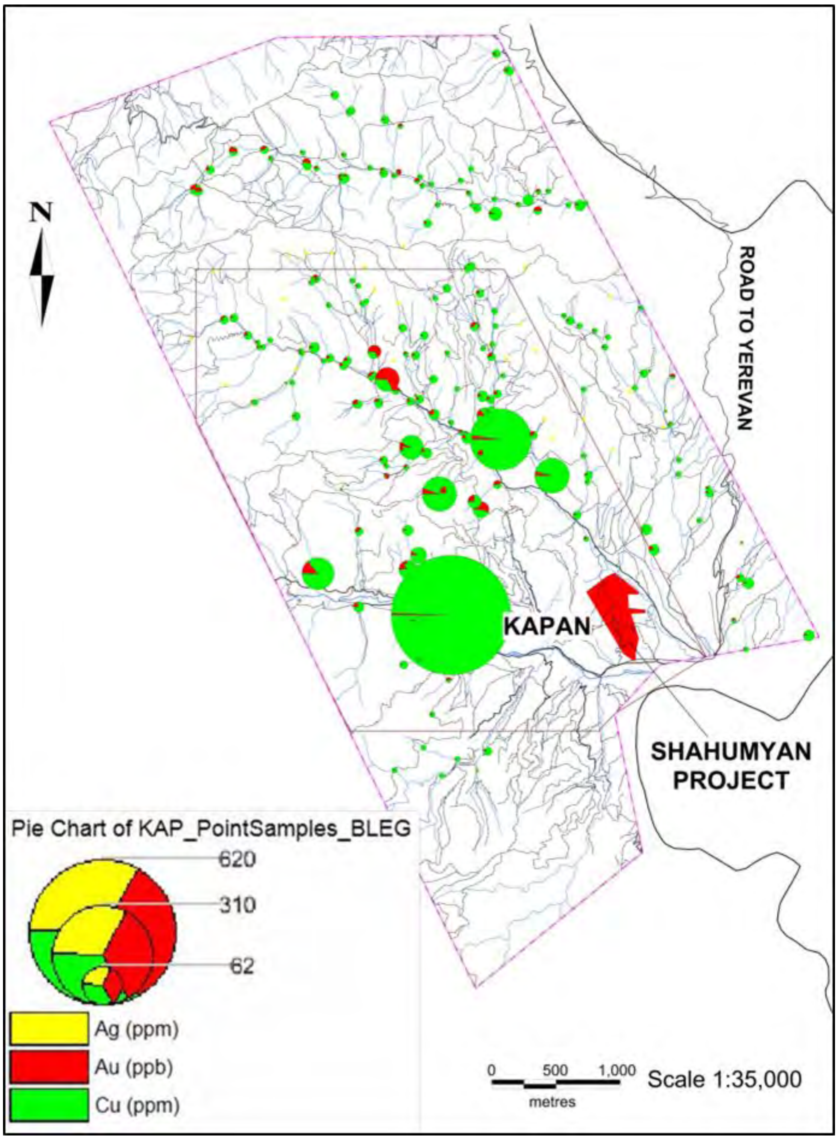


Figure 13. Stream sediment sampling and interpretation (DPMK,2010).

In 2013, soil sampling campaigns were conducted from May 2013 till December 2013

- Regional 200 x 200 m soil sampling was conducted within the western half of the exploration licence in areas where the prospective middle-Jurassic units were exposed for a total of 1,036 samples collected.
- Detailed 200 x 50 m prospect soil sampling in the Norashenik, Arajadzor, Gyandzebhut, Centralni South, Centralni east and Antarashat South prospects was undertaken, for a total of 2,260 samples collected.

Interpretation of these results is currently in progress.

9.6 Geophysical Target Follow-up

Geophysical anomalies identified in the Geotech VTEM-aeromagnetic survey that were considered having potential for mineralisation had follow-up work undertaken which included geological mapping/sampling and direct measurements of magnetic susceptibility. Geochemical sampling was also undertaken over the Kapan area in the most prospective zones. A total of 258 prospecting samples were collected for ICP analysis and 105 samples were submitted for alteration characterisation by Terraspec infrared mineral analysis. Field work follow-up of anomalous values indicated that a number of the anomalies are due to cultural activities (roads, villages, pipes, etc.). Further post processing is being completed to filter out these values and develop an appropriate exploration strategy.

9.7 Mapping

9.7.1 Review of Project Geology and Underground Mapping 2006

In 2006, Dr Brett Davis, then Group Structural Geologist for DPM, visited the Shahumyan mine with the aim of reviewing the geological model and paragenetic and structural history (Davis,2006). During the visit the majority of working faces underground were visited. His first pass paragenetic history is presented in Table 11.

A summary of observations listed in the report include:

- The Soviet vein interpretation used at the mine-site is strongly focused towards a single linked system of vein-bearing structures. Models do not take into account orientation data and there appears to be some discrepancy in the south of the Shahumyan where veins are modelled as dipping northwards as opposed to the more likely scenario of two separate veins dipping south.
- No geological reconciliation of the veins has been attempted to validate the interpretation/model.
- Vein-parallel structures are not shown on plans. In contrast to this, low-dipping structures are commonly seen but not included in the interpretation.
- Despite the relatively monotonous nature of the host rock there is sufficient variation in rock types to establish a working stratigraphy at the Shahumyan mine.
- Mineralisation appears to be restricted to units below the major roof thrust surface. Exploration should target this surface regionally.

9.7.2 Regional Geological and Structural Mapping- Jigsaw Geoscience - 2011

Regional 1:10,000 and 1:5,000 geology and structural mapping was conducted by the Jigsaw Geoscience Pty Ltd., over two campaigns during 2007 (Jigsaw,2007). Four QuickBird 1:10,000 images were utilised as a base for mapping and data was captured in MapInfo desktop GIS software. Point data and map grids used the Pulkovo 1942, Gauss Kruger Zone 8 coordinate system. All structural data was recorded in dip-dip direction format relative to magnetic north, unless stated otherwise. Deliverables provided include rock chip samples, photographs and a GIS database. The results of these tasks were used for development of a regional geological model.

In 2011 two additional mapping campaigns were undertaken by Jigsaw with the aim of extending the mapping completed in 2007 and to provide DPMK with an updated regional geological model. This structural and lithological interpretation from Jigsaw (2011) is illustrated in Figure 14.

9.7.3 Review of Geological Mapping by GeoMap - 2012

Geomap was engaged by DPMK to review previous geological maps and models generated by both historical Soviet geologists and more recently by Jigsaw (Geomap,2012). This review was focused on the Kapan area surrounding the Shahumyan deposit; an area of approximately 11x7 km. Geological traverses were conducted over the main mineralised area and data collected using a Discover Mobile v3.6 software package supported by a Trimble Nomad GPS with a built-in receiver. The mapping led to a new geological interpretation generated by modifying the Jigsaw shape files as these were the only digital geological model available.

This interpretation is presented in Section 7.3 (Local Geology).

Conclusions from this study were:

- Soviet maps are the most accurate with respect to rock type identification and accuracy.
- Stratigraphic relationships remain unresolved in all previous models.
- Mineralisation, scale and distribution of veins and alteration are consistent with the upper parts of a large porphyry style hydrothermal system of which Shahumyan represents the base mental carbonate zone with mineralisation developed in a series of tension arrays responding to strike slip movement on north-trending structures.
- Exploration at Shahumyan should target mineralisation down-dip of current veins.
- Centralni stockworks represent the lower transition parts of a high sulphidation epithermal zone with open pit potential.
- Abundant alteration at Barabatum indicates potential for currently unidentified mineralisation.
- There is potential for apron mineralisation in contact breccias and potential skarn mineralisation where the SSC-L limestone unit intersects the mineralisation system, most likely in the vicinity of the Centralni underground workings.

9.7.4 Centralni Copper Prospect

Seven kilometres to the west of the Shahumyan Mine is the Centralni copper Deposit. This deposit is a high sulphidation system which is characterised by east-west trending vein swarms and stockworks. The deposit was worked for approximately 150 years via underground and surface methods, closing in 2007 due to economic factors. A minimum of 30 Mt at 1.16%

copper grade has been extracted historically. It is DPMK's opinion that the Centralni deposit may still hold significant exploration potential in the form of untested, down-dip extensions of veins swarms and extensions to the deposit in the to the North and West.

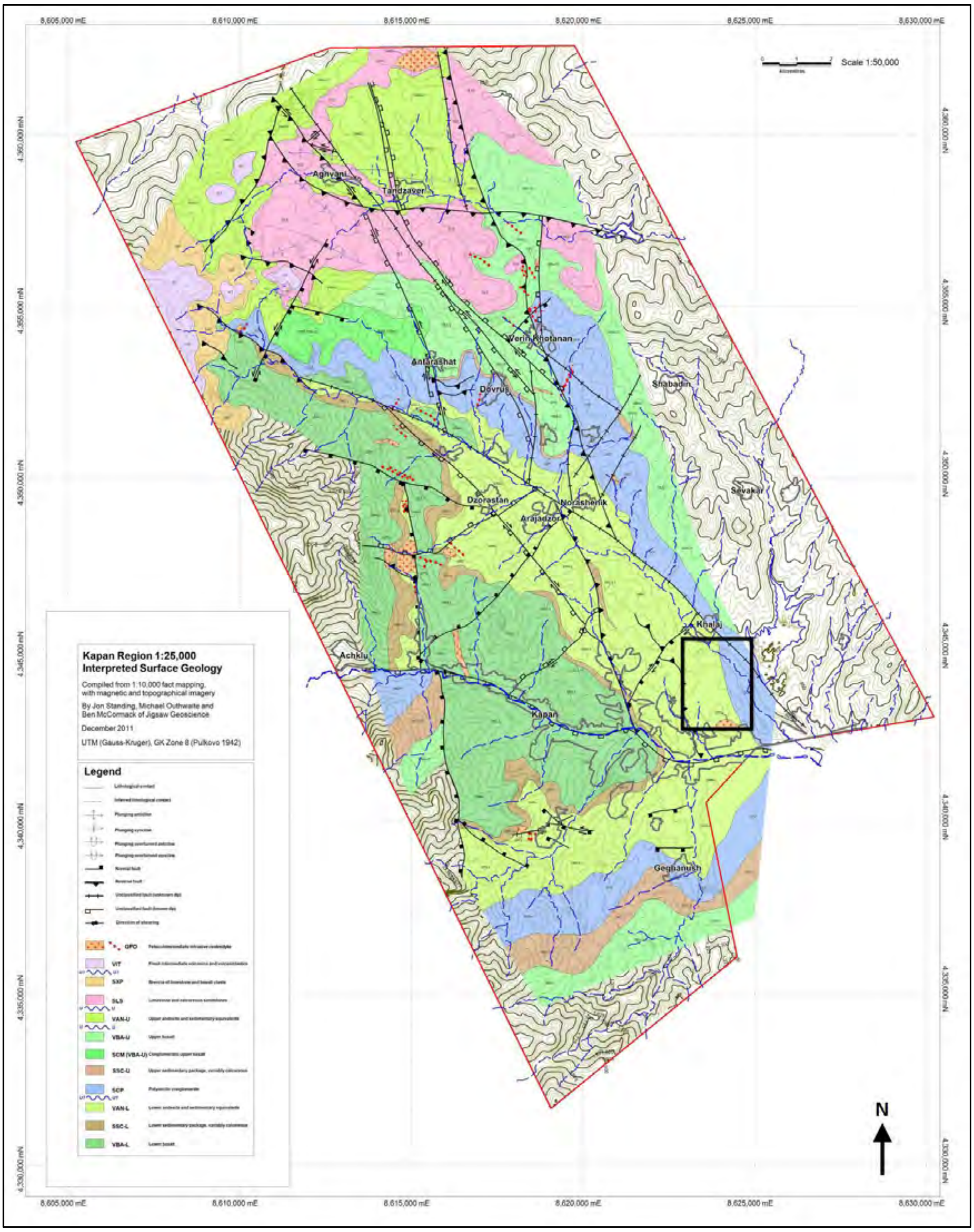


Figure 14. Jigsaw regional geological interpretation (Jigsaw,2011). Shahumyan mine location outlined in black, lower right of image.

9.8 2014 - Operational, near-mine and regional exploration programs

9.8.1 Operational resource development drilling

Currently DPMK's operational resource development drilling strategy combines resource definition drilling designed to a 30mx30m drilling grid with infill grade control holes. Wider spaced resource definition drilling is employed to re-categorise Inferred resources for long term mine planning. Whilst operational infill drilling has been designed to upgrade indicated resources to allow detailed production design and scheduling works. Drilling is planned chiefly from underground cuddies however a surface drilling program has also been scheduled to test veins which are too distant from development.

Table 14. Planned drill metres (DPM,2014)

Section	Location	Target veins	Metres Planned (m)
Central	Underground	22, 37, 105, 104, 102	6050
Central	Underground	30,34	1200
Central	Underground	34 zone	2260
North	Underground	9,13	5770
North	Underground	32,103,46	6700
North	Underground	35 zone , 5n	2400
North	Surface	3n, 44, 43	5000
North	Surface	5n, 46, 32, 35 zone	7600
South	Underground	26, 27, 28, 29, 31	6640
South	Underground	5s	1000
		Total resource development drilling:	**44,620 M**

DPM is committed to spend $2,400,000 USD for underground drilling and $885,000 USD for surface drilling.

9.8.2 Near-mine development

Surface drilling within the early portion of 2014 will focus on testing the possibility of extensions of the Shahumyan deposit within poorly explored areas of the near mine region. Plunge modelling of the main resource veins indicates that high-grade shoots appear to plunge moderately to the east, in-line with the dip of strata within the Shahumyan mine sequence. Using this basis, a short exploration drilling program has been planned to test conceptual targets.

During 2014 follow up work is planned to further understand the fault offsets in the mine area via detailed surface mapping and interpretations of UG and surface exposures. Major faults which cross-cut the Shahumyan deposit appear to have displaced mineralised zones. By resolving the faults movements, there may be extensions of mineralisation within the near mine area.

9.8.3 Regional exploration

Regional exploration programs during 2014 will involve prospecting and 1:2000 scale geological mapping of soil geochemical anomalies defined during 2013. Wide spaced soils sampling is also planned within remaining un-sampled portions of the exploration tenement. Defined anomalies will be subject to detailed geological mapping and possible trenching. Any proposed diamond drilling is contingent on the success of the target build-up work.

10 Drilling

10.1 Introduction

The database consists of both DPMK and Historical drilling data.

DPMK has carried out surface diamond drilling and RC drilling since July 2007. A total of 598 diamond drill holes and 102 RC drill holes have been completed for 181,309 m and 15,132 m respectively. DPMK has also undertaken approximately 176 underground diamond holes for a total of 57,802 m.

Historical drill data in the database includes 1,501 underground diamond drill holes totalling 277,025 m and 6,750 m of undifferentiated underground grade control data.

The available data, both historical and DPMK (and including channel sample and grade control data) is presented in Table 15 below.

Table 15. Summary of available historical and DPMK data.

Drill hole Type	Number of Holes	Total Metres
Surface Reverse Circulation (RC)	141	15,132
Surface Diamond (SD)	597	181,309
Underground Channel (UC)	6,853	22,816
Underground Diamond (UD)	176	57,802
Underground Grade Control (undiff.) (UDGC)	860	6,750
Underground Historical Channel (UHC)	5,820	21,573
Underground and Surface Historical Diamond (UHD)	1,485	270,952
DPMK Total	7,767	277,060
Historical Total	8,165	299,275
Total	15,932	576,335

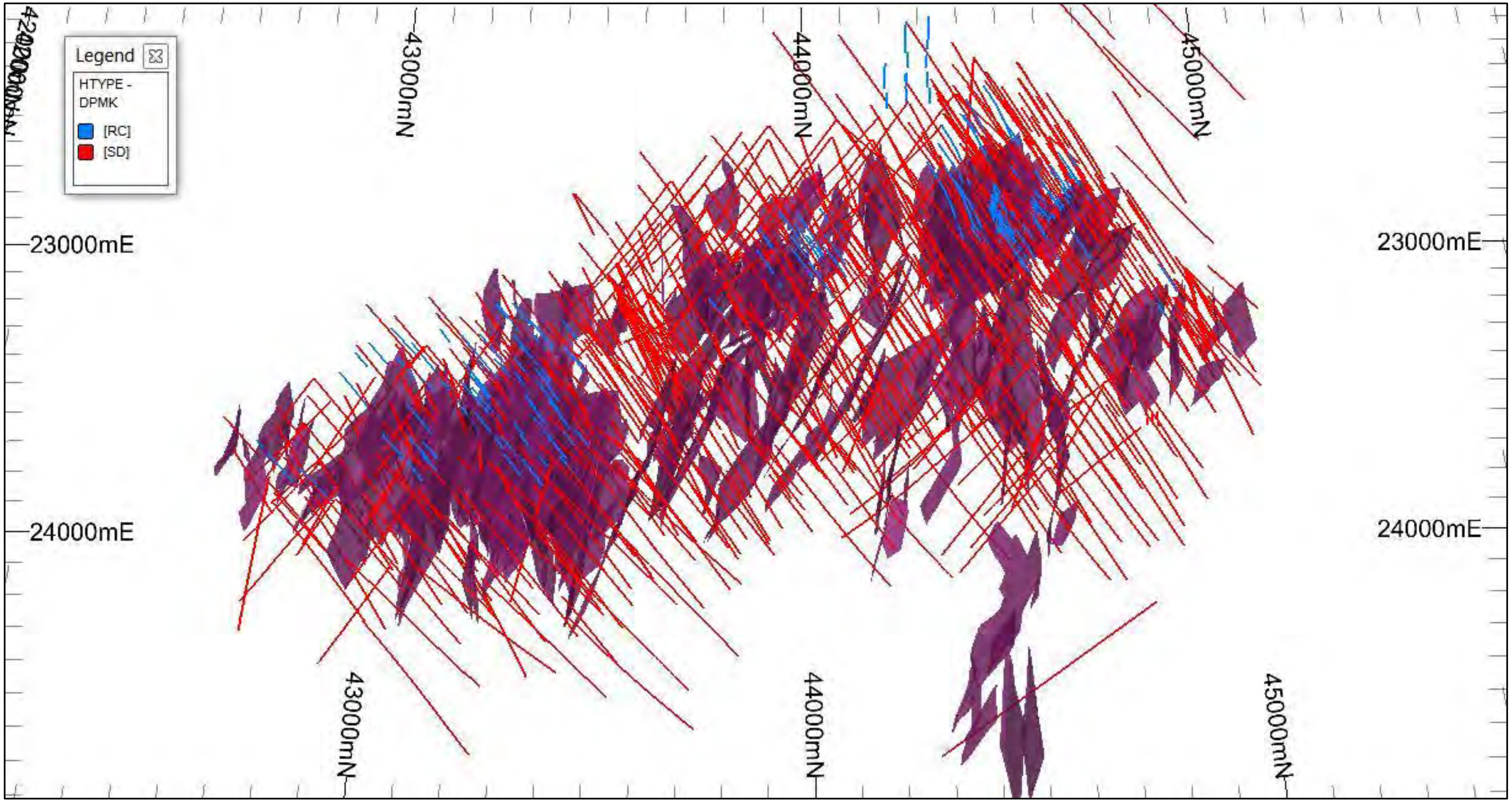


Figure 15. Surface holes available for the MRE. Oblique section, looking west. Vein Model is coloured pink.

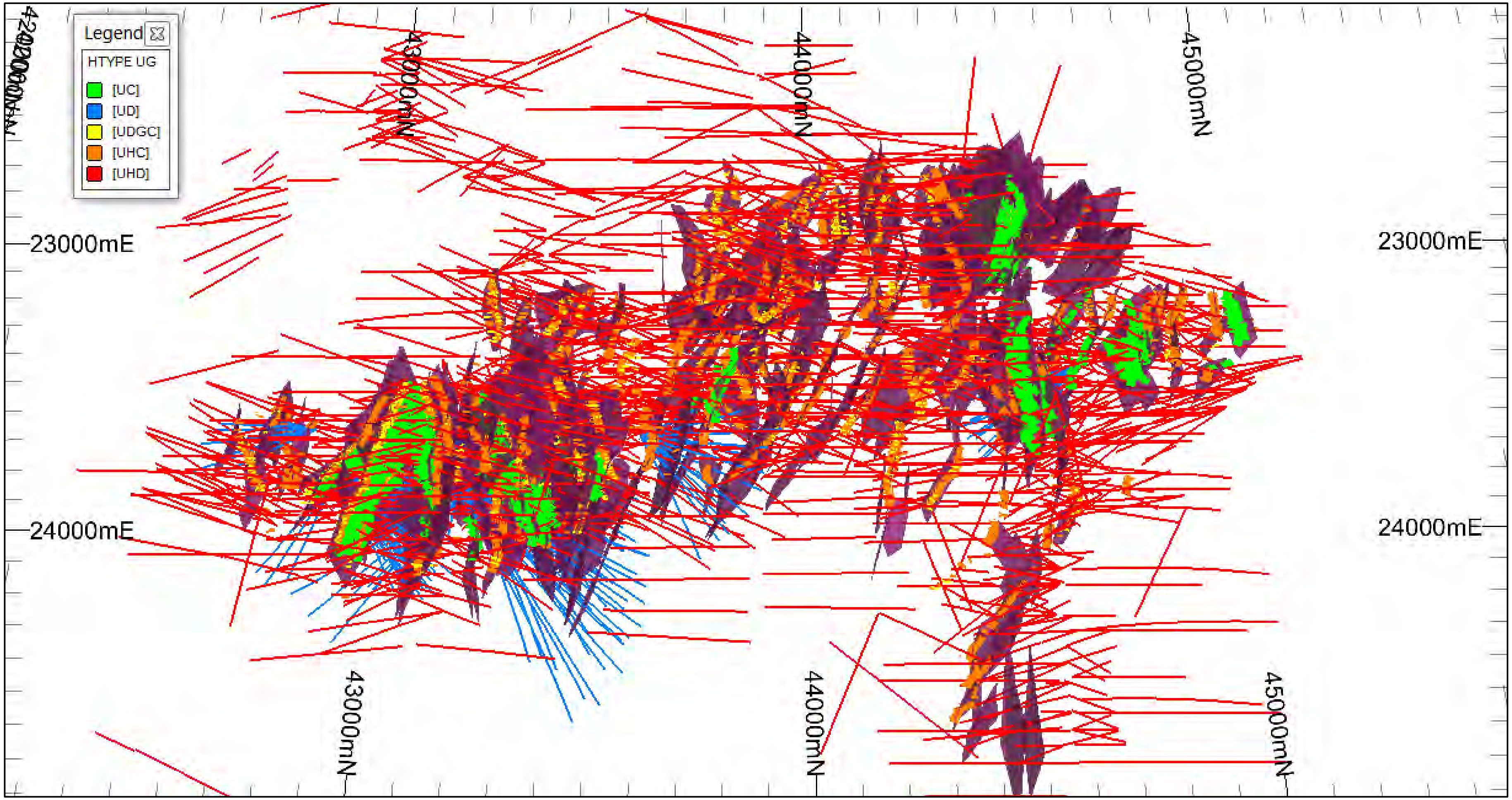


Figure 16. Underground data available for the current MRE. Oblique Section looking west. Vein Model is coloured pink.

10.2 DPMK Drilling

10.2.1 Surface - Diamond Drilling

DPMK has carried out surface diamond drilling (DD) at the Shahumyan deposit from July 2007. In summary:

- Diamond Drilling procedures pertain to both surface and underground grade control drilling.
- Typically, surface DD holes were collared with PQ rods, with HQ commencing at the base of the transported clay material (approximately 10-30 m vertical thickness) until the end of hole.
- A relatively small amount of drilling was completed with conventional NQ and HQ, mainly in late 2007 to early 2008 and within the central parts of the Shahumyan Mine Concession.
- CSA considers that there are no drilling, sampling or recovery factors that could materially impact on the reliability of the drilling results.

A total of 597 diamond drill holes have been completed for 181,309 m, with an average hole depth of 304 m.

Figure 17. Surface Diamond Drilling at Shahumyan

10.2.2 Surface - Reverse Circulation Drilling

Reverse Circulation (“RC”) drilling at Shahumyan has been conducted on a 20 mE x 20 mN grid for resource definition purposes. In summary:

- Water is initially injected to facilitate drilling to the base of the transported sedimentary clays and the base of this layer is generally very sharp. During this part of the drill hole, the cyclone is manually cleaned every rod (6 m).
- Rigs are fitted with dust extractors and an automatic cyclone system comprised of vibrating components coupled with air jets.

A total of 141 RC drill holes have been completed for a total of 15,132 m, with an average depth of 107 m.

Figure 18. Surface RC Drilling at Shahumyan

10.2.3 Underground - Diamond Drilling

Underground drilling follows the same procedure as surface diamond drilling. Underground drilling currently uses either BQ or NQ drill diameter. Specifically:

- NQ holes are drilled by Boart Longyear rigs.
- BQ/HQ holes are drilled by Diamec 262 rigs.

Underground drilling outlines areas of the veins and determines possible areas where wider stoping / different style of mining would be possible.

DPMK have undertaken approximately 176 underground diamond holes for a total meterage of 57,802 m.

10.2.4 Diamond Drilling - Geological Logging

Logging procedures pertain to both to surface and underground diamond drilling.

In summary:

- Inner tube splits and core lifters are washed prior to reuse in successive drill runs.
- Wooden core blocks are placed between runs, recording the length of the run and/or core loss.
- Forced breaks made by the drillers are marked on the core at both sides of the break using a red cross.
- The plastic core trays are clearly labelled with the hole ID and depth (from-to), tray number. The top left hand corner of the edge of the tray is labelled "START".
- All drill core is transported to the secure core facility on a daily basis, to be marked-up. Core is covered using tray lids during transit to prevent core shifting place within the trays.
- Drill core is photographed both wet and dry by a semi-automated digital system.
- Diamond core recovery is measured during the core mark-up process, prior to logging and cutting.
- Diamond drill core logging is undertaken at a minimum of 25 cm intervals. Data is captured within digital spread sheets for geological logging, geotechnical logging, structures and sampling intervals. Core is logged directly onto Tablet computers using Field Marshall, a Micromine interface.
- Summary geotechnical logging consists of recording RQD, with core loss being recoded as 0 RQD. Structural logging is undertaken by recoding the alpha and beta angles of at least one structure per metre of core.
- The use of tablet computers follows appropriate practices, allowing for consistent logging using deposit specific geological logging codes. The presence of type lithology and type alteration boards supports good logging practice, and allows for consistent training of new staff.

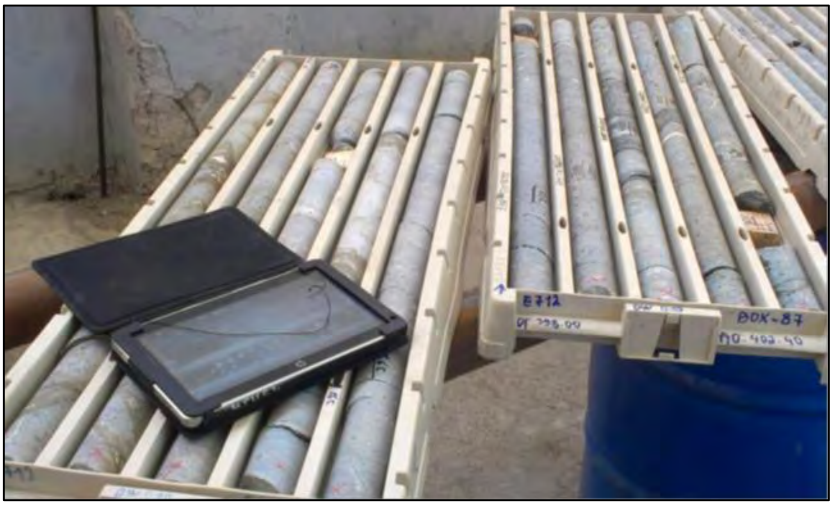


Figure 19. Data collection from core is input directly into HP Toughbooks for upload to acQuire

CSA considers the procedure described above as adequate; however a review of the dataset showed that there is insufficient detail in the logging to classify mineralisation based on lithcode, and improvements are required in this area as refinements to the geological model are made in the future. This has been discussed in more detail in Section 12.2.

10.2.5 RC Drilling - Geological Logging

Logging codes and libraries follow that of the diamond core logging described in the previous sections. Other procedures include:

- During logging of RC chips a representative portion of the cuttings, from each successive metre, are sieved to remove any fines and stored in labelled chip trays. Geological information is then recorded from these chips.
- DPMK procedures state that logging is carried out per metre, paying particular attention to oxidation type, rock type, Tectonic/structural fabric, veining/intensity, alteration/intensity, sulphides/intensity and moisture content.
- Instances of voids, cavities or insufficient samples are recorded.

Figure 20. Sieved RC chips collected for metre intervals drilled and available for logging by the geologist.

10.3 Pre-DPMK Drilling

10.3.1 Data Sources

Historical data (including drill, grade control and channel sample data) have been compiled from four principal sources, these are:

- The Geological Fund of RA (Republic of Armenia) in Yerevan
- The reports of the Kapan mine
- International GIS geology sources
- Historic Reports.

For the historical reports: more than thirty historic reports have been located and the pertinent data scanned, translated, digitised and georeferenced. These reports were sourced from:

- The geological prospecting expeditions of ArmGeo
- The expeditions of Ministry of Precious Metals
- The Institute of Geology of Armenia, TSNIGRI and other organizations.

All historical hard copy data was entered manually into data entry software and then imported into acQuire database management software.

Graphical documentation was captured digitally using a GTCO-Calcomp roll-up digitiser system and scanned using a Colortrac CX-40 scanner.

The main types of data captured were:

- Drill hole logs from historic resource definition drilling from the 1960's until 1990's
- 1:1000 scale cross sections
- 1:200 scale sample plans with detailed structural mapping
- 1:1000 level summary plans with development and survey stations
- Vein Projections detailing historic resource calculations.

The following checks were undertaken to verify the process of digital data capture:

- After de-surveying of the digitised down-hole survey data in a mine planning software, the position was validated against available from historic plans, sections and drill hole catalogues to determine the accuracy of the entered data.
- Spatially, historic drill hole logs match very well with plans and sections and there was no need to correct collar positions.
- Entry of historic assay values was also deemed accurate after checking recorded values in mineralised zones with the same data used in historic polygonal calculations.

Figure 21. Digitiser being used to capture plans

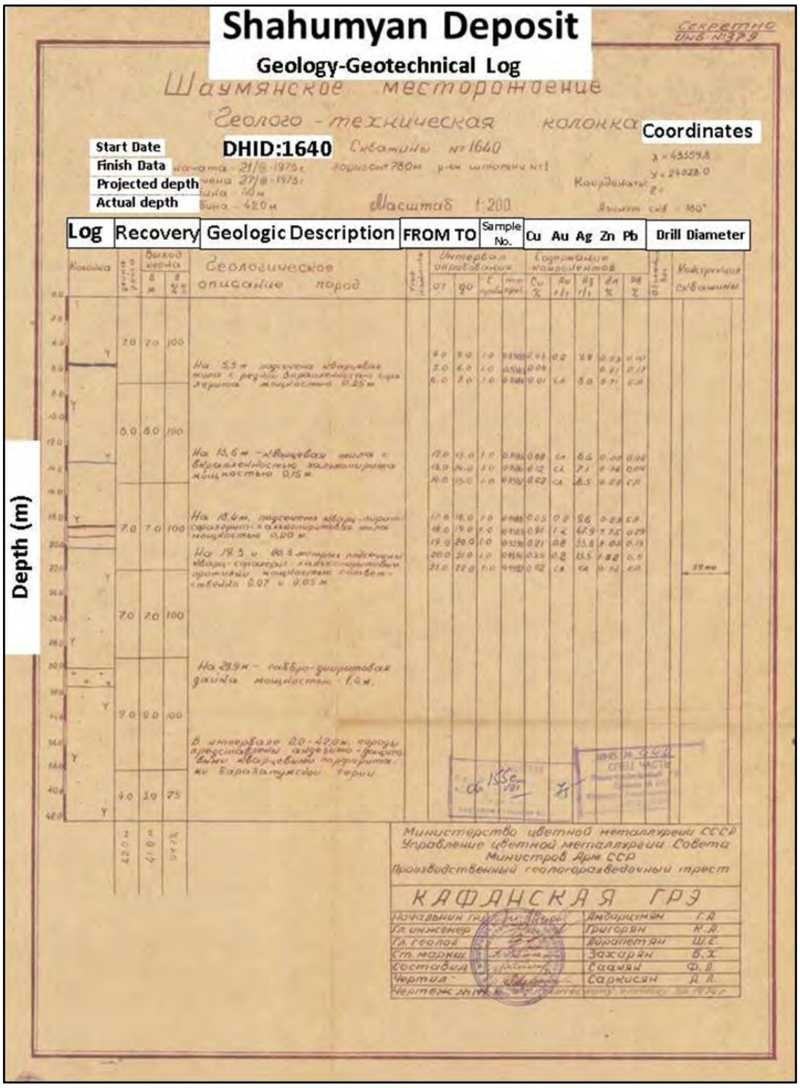


Figure 22. Geological Log from Soviet exploration party with key elements translated.

10.3.2 Underground - Diamond Drilling

Drill rigs were conventional in fashion, meaning drill core was retrieved from the core barrel by pulling rod strings to surface as opposed to modern wireline techniques.

A total of 1,482 historical underground diamond drill holes for a total of 275,920 m, with an average depth of 186 m are contained in the database.

Drilling was undertaken primarily using Russian made, electric powered SKB 4 and ZIF 500 rigs.

Early underground drilling typically comprised of horizontal drill holes up to 450 m in length on a regular grid across the deposit. As the deposit was developed via a network of shafts and adits, underground drilling was used to define resources using positively and negatively inclined holes typically to 200 m in length and either in north or south azimuths aiming to delineate east-west trending mineralisation. Below the 700 development level, underground fan drilling was frequently employed to target the deeper portions of the deposit (600, 500 and 400 levels).

10.3.3 Underground – Grade Control Drilling

Underground grade control drilling was undertaken with a small, man-portable Kempe drill rig, capable of drilling a length of 50 m, with a bit diameter of 42 mm.

These "stab holes" were taken typically every 10 m to a maximum spacing of 20 m within areas of visible mineralisation, predominantly within portions of the deposit with bigger veins, with notable alteration haloes.

These UDGC holes were digitised off historic sections and now appear as a single hole in the database with a missing interval in the centre which corresponds to the drive itself and is often channel sampled.

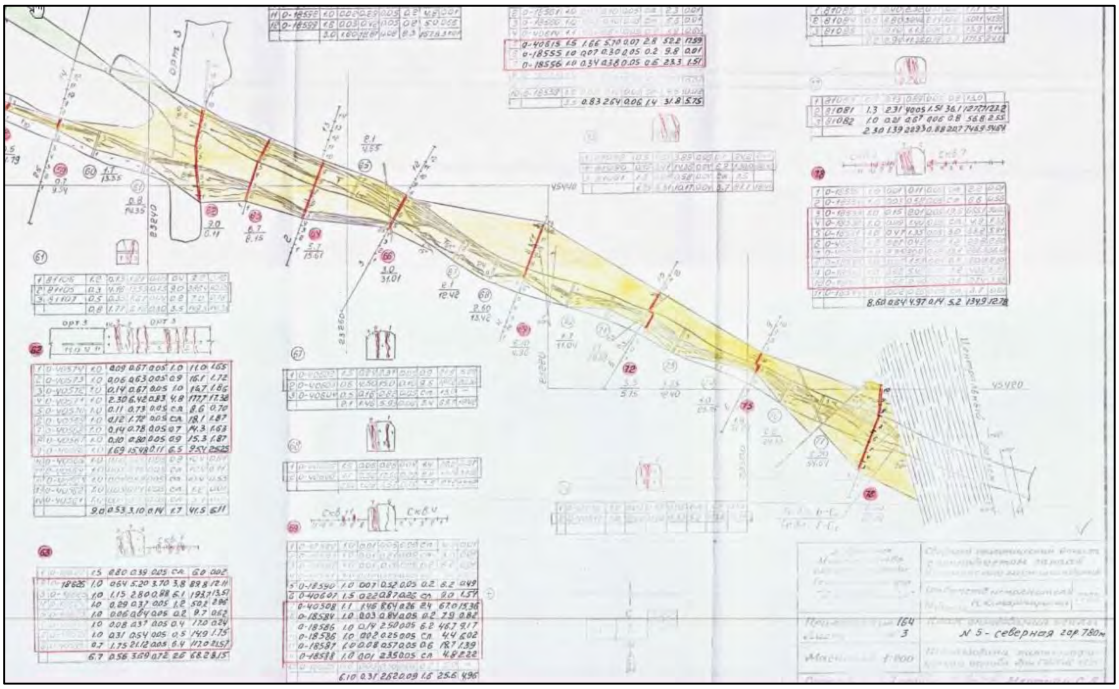


Figure 23. 1:200 plan of underground development showing location of face samples and grade control holes which are digitised from the plan

10.3.4 Diamond - Geological Logging

Historically, core drilled were logged either underground or on surface in a logging facility. Geologic logs were created primarily by using a graphical schematic strip log with lithology, mineralogy and structural annotations added.

Core descriptions recorded lithology, texture, alteration and mineralisation style however historic lithological logging was descriptive and not standardised and therefore currently no geologic descriptions have been translated and entered into a database.

10.4 DPMK Surveying

10.4.1 Grid Control

In late 2006, prior to resource drilling, DPMK engaged Spectrum Survey and Mapping of Perth, Western Australia, to investigate the Gauss Kruger – Pulkovo 1942 Zone 8 projection (“GK8”) coordinate system being used at Shahumyan and to introduce a Real Time Kinematic (“RTK”) GPS system to the project (Spectrum 2006). All reporting to Armenian government departments must be in the GK8 system.

During this review the values of the existing survey control network relative to the GK8 system were checked which involved constructing a GPS survey base station point (“Office”) on the roof of the DPMK Mining office in Shahumyan. This location of this point was then measured by GPS (7 hours static observed data). The measurement is taken in WGS84 coordinate system and converted to GK8. The point “Office” has also been surveyed from the Armenian government supplied survey station Niva1 and compared to the GPS result. There is a significant discrepancy between the 2 measurements and this was attributed to inaccuracies in the old government network caused by the out-dated methods at the time the previous network was established. The discrepancy has been noted and the true GPS measurement has been adopted as true. This adopted baseline for this GPS network and all future work on this project is the government point “Niva1” and a DPMK control point “A2” with Niva1 being the origin. These points were selected because they are accurate and part of the existing network and are in good proximity to most mining works. A total of 13 survey control points have been derived from static data and subsequently converted to Gauss Kruger, Pulkovo 1942 Zone 8, as presented in Table 16.

Table 16. Survey Station Control Points (local mine grid)

Station	Easting	Northing	Elevation
CA1_1	23914	42508	781.031
A2	23925	42718	854.999
MPILLAR1	24029	42423	776.591
MPILLAR2	23890	42472	779.972
N1	24515	44884	842.815
SHM01	23858	45178	791.924
WM1	23903	42384	762.362
WM2	23888	42387	761.159
WM3	23872	42383	761.418
WM4	23867	42360	761.101
WM5	23846	42344	759.965

The implementation of the RTK system has allowed accurate GPS static surveys to be carried out and produces accurate survey control results particularly in of areas rugged topography.

10.4.2 Surface - Collar and Location Survey Control

All surface drill holes were located and collared following drilling using a GPS Topcon HIPER GA tools with an accuracy of 3 mm - 10 mm. For engineering works and stability monitoring a Total Station (Leica TPS 1203) was used with an accuracy of about ±3-5 mm.

10.4.2.1 Survey QA/QC

Surface surveying, for topography, hole mark out or collar pick up always begins with the setup of a base station on a known survey point. This base station is then linked to the roving receiver, which is used to pick up or mark out points. A check of 2 known base stations is undertaken at the end of each survey, as a control.

CSA considers this procedure to be adequate and appropriate.

Figure 24. GPS Base Receiver (top) and GPS Rover Receiver (bottom) used for collar surveying

10.4.3 Surface - Hole Orientation Surveying

Down-hole surveying is undertaken by gyroscope, manufactured by Stockholm Precision Tools AB of Sweden. This tool is regularly calibrated and error is ±1% for azimuth. Additionally, a backup EMS Ranger tool is maintained on site.

RC holes are surveyed either within rods or within open hole at 25 m intervals. Diamond holes are surveyed at 25 m in open holes or at 50 m intervals if surveyed during rod pulls.

Survey results are checked by a project geologist regularly. Unexplained deviation greater than 0.20 per metre, either from true hole deviation or magnetic interference, were investigated during drilling.

CSA considers this procedure to be adequate and appropriate.

10.4.4 Underground - Collar and Face Survey Control

Underground surveying utilises a Leica TC 15a total station setup and an Optech Cavity Monitoring System (“CMS”) for void modelling, with a small Ranger Survey tool for long hole stoping drill hole checking. Summary points for the submitted Underground Survey SOP are:

- Wall stations are based on a TRILATERATION (resection) method of measurement and calculation.
- Wall stations are no more than a maximum of 40 m apart.
- Wall stations are located with sufficient distance from the working face so not to be compromised by blasting.
- Station set-up is to be approximately 1.5 m – 2.0 m from the nearest target. If closer than 1.0 metres to the prism the Auto Target Range will limit reading accuracy.
- Maximum azimuth residual of set-up point should be no greater than 00 00’10”.
- A gyro check should be conducted first on initial datum wall stations and compared to mine survey datum observation for comparison.

10.4.4.1 Survey QA/QC

- All underground headings are surveyed for advance every fourth day. Twice a year a control traverse of each level of the mine is undertaken.
- The totals stations are calibrated on site every 3 months. Every 6 months they are sent to the manufacturer to undergo maintenance and calibration; both stations are currently under warranty.
- The Optech V400 undergoes yearly maintenance and calibration by the manufacturer.

CSA considers this procedure to be adequate and appropriate.

10.4.5 Underground - Hole Orientation Surveying

Underground drill holes are surveyed every 30 m during rod pulls using the Reflex EZ-Trac camera system.

Production drill holes are down-hole surveyed using a Ranger Mask 2 multi/single shot system, DPMK reports that 20% of drilled production holes are surveyed.

Surveys are checked by project geologists for any signs of deviation.

The Standard Operating procedure submitted by DPMK indicate that the QA/QC for the Ranger Tool is 1 calibration check per week using a frame, illustrated in Figure 25 set at a defined inclination (-29.9°). The azimuth of the frame is rotated 4 times and three inclination readings taken each time to confirm consistency of the instrument.

Survey tools are annually calibrated at Reflex Instrument's European office.

CSA considers this procedure to be adequate and appropriate.

Figure 25. Frame used to hold Ranger Tool for weekly calibration check

10.4.6 Topography

The topography utilised at the project comprises a combination of several levels of accuracy:

- High accuracy RTK DGPS covering the Shahumyan mine site and some peripheral areas.
- Total station pick up for areas which were too steep to safely traverse.

- Historic 1:5,000 contours.
- Historic 1:10,000 contours.
- Helicopter born geophysics radar-altimeter 80 m spaced lines has been used to construct the remainder of the licence area.

10.5 Pre-DPMK Surveying

10.5.1 Collar Survey Control

Hole surveys were likely undertaken using appropriate optical methods, using theodolites and survey loops. Collars for historical holes were ultimately digitised from underground hard copy plans. Collars and traces were desurveyed in a mining package and plotted positions checked back against the original plans.

10.5.2 Hole Orientation Surveying

DPMK located 563 hardcopy downhole survey reports in various conditions for the approx. 1,500 underground historic drillholes in the Shahumyan deposit. After validating the data statistically and visually it was added into the onsite Acquire database. The drillhole logs themselves were taken from the Geofont in Yerevan, a library of historic geologic expedition data. These logs themselves were scanned and entered manually into excel by the DPMK exploration database team.

Of the 563 downhole survey reports, 34 were considered unreliable due to deviations from other references (plans/sections/drillhole logs/historic reports). The vast majority of the remaining reports are typically 'head and tail' surveys with shots taken near the collar (i.e. at approx. 10 m depth) and at the end of hole. Approximately one third of the reports have regular 25 m sample shots.

Many of the reports do not have an azimuth value at the first shot, for these holes the next available azimuth value was used. This is not ideal as it leads to almost linear drill traces. Also different reports were surveyed using different instruments at different periods of exploration. As a result, some logs show zenith angles instead of standard dip angles measured from a horizontal plane. DPMK corrected this issue prior to export of the final data.

DPMK believe that further reports may be available at the Geofont in Yerevan, which may be recovered in the future (Overall, R.,pers comm).

CSA believe that the new downhole survey data is an improvement relative to the previous dataset used in the MRE, although lack of initial azimuths at 0 m, lack of regular shots downhole and incomplete datasets are not an ideal.

11 Sample Preparation, Analysis and Security

11.1 DPMK - Sampling

11.1.1 Diamond Drilling – Sampling

Diamond drilling sampling procedures pertains both to surface and underground grade control drilling.

All drill core is sampled following transportation of the core to the core yard facility and after all geological and geotechnical logging and photographing.

The core yard facility is well organized and well lit, as illustrated in Figure 26. Sampling procedures are as follows:

- Sample intervals were typically 1 m in barren zones and a minimum of 25 cm, maximum of 1 m in mineralised zones and are to honour geological contacts. Samplers are advised to estimate true thickness from the alpha angle.
- A cutting line was drawn parallel to the orientation line or, where this was not available. Core was cut with a diamond saw with care being taken to retain the orientation line.
- Drill core was routinely sampled on the right hand side.
- Half core samples were bagged with unique sample tags and submitted for laboratory analysis with the residual half core retained in the plastic core trays.
- Sample weights varied between 2 and 7 kg depending on length of the sampled interval, with an average weight of approximately 3.5 kg.

Figure 26. A photo of the core yard.

11.1.2 RC Drilling - Sampling

RC sampling adheres to the following procedure:

- Holes were drilled using a sampling bit of 5 ¼” diameter.
- RC samples were collected at metre intervals from the drill rig cyclone using an air operated system.
- Samples were collected down hole using a face sampling bit.
- The sample lot was passed through a riffle splitter, to obtain a 2 kg sub sample and a reject sample.
- Contamination was prevented by:
  - The rods being pulled back and flushed with clean air every 1 m.
  - The cyclone was checked for sample build up during each rod pull (6 m). The riffle splitter was cleaned between each sample using a rubber hammer and compressed air.
  - The weight of the reject and sub sample were routinely recorded, to monitor sample recovery.
- Sub-samples were collected in polythene bags marked with hole ID, metres from and metres to.
- The sampler inserted a pre-numbered sample ticket into each bag, which was then sealed at the rig.
- Historically, when damp or clayey samples were encountered, samples were collected using a polypipe spear. Multiple spear samples were collected throughout the sample lot, to make up a 2 kg sub-sample.
- Current DPMK policy is to halt drilling whenever unmanageable water is encountered. The hole is then completed using a Diamond drill core tail.

CSA considers this procedure to be adequate and appropriate.

Figure 27. RC Samples lined up at the rig with riffle splitter and cyclone located in the background

11.1.3 Underground Channel – Sampling

Underground channel sampling was reviewed onsite during a site visit by CSA Principle Mine Geologist Steve Rose. The DPMK channel sampling procedure is as follows:

- Face samples are taken by painting up the vein on the face, and then painting sample lines. The sample line is at chest height (1.3 m).
- Samples are a minimum of 0.15 m long and a maximum of 1.5 m. Samples break on lithology.
- The samples are taken by chipping the rock face using a Hilti hammer drill fitted with a chisel blade over the width of the panel (10 cm) and along the length of the sample line. The entire face is sampled.
- The sample weights are about 4 kg in size, collected into calico bags using a metal sample hoop. Sampling is a two man operation, with one sampler using the hammer drill, and the other sampler holding the sample hoop. This means that the sample is collected directly.
- The face location was measured using either a Disto instrument or a tape measure, using a survey wall station as control.

Figure 28. Photo of the face at 703L Vein 3 West, showing sample marks

11.1.4 QA/QC Samples

After geological logging, core sampling and density sampling the insertion of QA/QC samples is undertaken. Site standard QA/QC procedures are as follows:

Insertion of standards and blanks into the sample stream in mineralised zones is supplementary to those placed in undifferentiated material.

If the location of a field duplicate is near a mineralised zone, it is permissible to move the location of the field duplicate to test a vein instead.

In addition, the lab is provided with sample numbers that require specific barren flushes, corresponding to down hole of samples suspected of containing high grade mineralisation.

Frequency of QA/QC sample insertion is presented in Table 17 and Table 18.

Table 17. QA/QC insertion for barren material

Frequency in undifferentiated material	Frequency of insertion
Standards	1 in 20
Field Duplicates	1 in 50
Blanks (Coarse or Fine)	Beginning of hole and every 50

Table 18. QA/QC insertion for mineralised material

Frequency in mineralised areas	Frequency of insertion
Standards	After Mineralised zone
Field Duplicates	Only in 1 m samples
Blanks (Coarse or Fine)	After Mineralised Zone

11.1.5 Bulk Density Sampling

There are 49,626 bulk density samples in the DPMK underground diamond and surface diamond datasets.

As a standard procedure, an approximate 15 cm billet is taken every 20 m from an area representative of the 20 m interval. Additional samples should be taken where lithologies vary.

Any veins greater than 0.5 m are sampled along within the hanging wall and footwalls at approximately 1 m either side.

Bulk density was measured using the Archimedes principal, with the following procedure:

- 10 cm sample of core was dried in an oven for 4 hrs @ 600 C
- Sample was weighed to ascertain "dry weight"
- Sample immersed in paraffin wax
- Sample was re-weighed, "weight of wax" was calculated
- The sample was then suspended and weighed in water to determine the volume.

Bulk density was calculated in the following way:

- Bulk density = (mass core)/ (mass in air – mass in water)-(mass wax/0.93)

CSA considers the procedure described above as adequate and of an appropriate standard, however the following observations are made:

CSA reviewed the density data, this is described in more detail in Section 12.3.

11.2 DPMK - Sample Analysis

11.2.1 Laboratory Sample Preparation

The following sample preparation is undertaken at the SGS laboratory on site. The SGS facility in Kapan is independently managed by SGS management under internal procedures and international accreditation. Although DPM are the facility's major client, and at times DPM staff have been employed at this facility, CSA considers there is no conflict of interest of any issue of independence.

Procedures are as follows:

- All samples were dried for a minimum of 12 hours at 1050 °C.
- Diamond core samples were disaggregated and jaw crushed to a nominal -6 mm samples larger than 4.5 kg were riffle split following crushing.
- RC samples were pulverised following drying.
- All samples were riffle split to 2 kgs.
- Samples were then pulverised to a nominal 95% passing -75 µm, using an LM5 (the lab performed a sieve test on every 20th sample). The bowl and puck of the pulveriser were routinely cleaned using a barren flush (every 20th sample).
- Barren flush material was inserted following any samples identified with visible gold, or high grade material.
- 3 pulp samples were collected one for Analysis, umpire analysis and storage.
- Pulps that were sent for analysis were packed in a box, with standards inserted; the box was then sealed and sent to the assay laboratory.

11.2.2 Laboratory Assay Methods

Sample analysis is undertaken at one of 4 independent laboratories:

- SGS Welshpool (Australia).
- SGS Tianjin (China).
- SGS Chelopech (Bulgaria).
- SGS Kapan (Kapan).

However, the majority of DPMK analyses are undertaken at SGS Kapan or SGS Chelopech.

Analytical methods at Chelopech are detailed below:

- Gold: was analysed by either a 25 g or 50 g fire assay with AAS finish.
- Silver, lead, zinc, copper, arsenic, mercury and bismuth were analysed by 0.3 g aqua regia digest with AAS finish for high grade samples (Chelopech) with low grade samples analysed by ICP-OES (Tianjin and Welshpool).
- For each batch of 40 samples the laboratory inserted 4 replicates, 3 second splits, 3 CRMs and a blank sample, for QA/QC purposes.

11.2.3 Laboratory Accreditation

According to DPMK personnel, the SGS operated lab for Shahumyan operates under SGS accredited procedures and to SGS standards. It does not however have full ISO 17025 accreditation.

SGS attempts to counters this lack of accreditation by applying their ISO approved procedures and management certificates. SGS state that they also participate in internal (SGS) round robins, and take part in the international competition organized by Geostats Australia.

All SGS methods use certified reference material standards, blanks, repeats, and duplicates inserted. Also samples are submitted to other labs for QA/QC checks overseas.

For Welshpool, Tianjin and Romania the SGS laboratories are fully accredited labs to ISO 17025 standards.

11.2.4 Sample Security

All samples are securely transported by DPMK Staff from the drill rigs to the sample preparation facility. With regard to sample preparation and assaying at Shahumyan, the SGS managed analytical laboratory is located within the confines of the DPMK compound, access to which is secured by locked gate and 24 hour guard. Samples are prepared and processed in this SGS facility to produce sachets of pulped material which were packed into sealed cardboard boxes and sent to SGS Chelopech (Bulgaria), SGS Tianjin (China) or SGS Welshpool (Australia) for final analysis.

Reference material is retained and stored at the DPMK compound, as well as chips derived from RC drilling, half-core and photographs generated by diamond drilling, and duplicate pulps (SGS prep lab) and residues of all submitted samples.

CSA have visited this faciliaty and observed and reviewed all steps in the security procedure and chain of custody. CSA considers sample security to be adequate.

11.3 DPMK - Assay QA/QC

11.3.1 Introduction

A quality assurance and quality control programme has been implemented by Dundee Precious Metals in an attempt to provide adequate confidence that sample and assay data can be used in resource estimation. The most recent assay QA/QC review indicates that the system in place is sufficient to assess the data reliability, accuracy and precision. This QA/QC review relates to analyses undertaken at SGS Kapan for the period 1 February 2013 to 30 September 2013 for gold, copper and zinc analyses.

Note: DPMK only inserted QA/QC material in the drilling samples (surface and underground). However there is laboratory inserted QA/QC material included with the underground channel samples as well as the drill samples. This QA/QC review includes the DPMK and lab QA/QC material but no historical QA/QC data is available.

11.3.2 Laboratories Included in QA/QC Review

Samples were analysed at the Kapan onsite laboratory managed by SGS.

11.3.3 *Certified Reference Materials (CRMs) – 'Standards' and 'Blanks'*

DPMK submitted a blank (Blank_RSD), project specific certified reference materials (DGS07-02, DGS07-03 and DGS07-05) as well as gold standards for analysis with their samples. Various gold, base metal and sulphur standards were included and analysed by the laboratories. Expected values and ranges for these standards are presented in Table 16. The expected minimum and maximum values are twice the standard deviation from the expected value.

Blank_RSD is an Australian quartz sand, obtained from the laboratory where it is used to clean the pulverisers.

Table 19. CRM's used in Sampling

StandardID	Element	Units	ExpectedValue	ExpectedMin	ExpectedMax	ExpectedStDev
DGS07-02	Ag	ppm	5.4	4.28	6.52	0.56
DGS07-02	Au	ppm	0.31	0.256	0.364	0.027
DGS07-02	Cu	ppm	594	516	672	39
DGS07-02	Pb	ppm	156	130	182	13
DGS07-02	Zn	ppm	3066	2712	3420	177
DGS07-03	Ag	ppm	12.6	10.34	14.86	1.13
DGS07-03	Au	ppm	0.56	0.502	0.618	0.029
DGS07-03	Cu	ppm	1480	1324	1636	78
DGS07-03	Pb	ppm	421	357	485	32
DGS07-03	Zn	ppm	8428	7750	9106	339
DGS07-05	Ag	ppm	142.3	121.78	162.82	10.26
DGS07-05	Au	ppm	7.16	6.67	7.65	0.245
DGS07-05	Pb	ppm	4942	4260	5624	341
DGS07-05	Zn	ppm	60332	57018	63646	1657
GBM396-10	Ag	ppm	11.6	10.2	13	0.7
GBM396-10	Cu	ppm	2897	2549	3245	174
GBM396-10	Zn	ppm	10601	9067	12135	767
GBM399-1	Ag	ppm	1.1	0.3	1.9	0.4
GBM399-1	Cu	ppm	86	72	100	7
GBM399-1	Pb	ppm	125	87	163	19
GBM399-1	Zn	ppm	97	79	115	9
GBM901-5	Ag	ppm	6	5	7	0.5
GBM901-5	Cu	ppm	1520	1302	1738	109
GBM901-5	Pb	ppm	521	443	599	39
GBM901-5	Zn	ppm	5189	4725	5653	232
GBM999-4	Ag	ppm	98.4	83	113.8	7.7
GBM999-4	Cu	ppm	1029	903	1155	63
GBM999-4	Zn	ppm	221	191	251	15
GBMS304-6	Ag	ppm	6.1	4.5	7.7	0.8

StandardID	Element	Units	ExpectedValue	ExpectedMin	ExpectedMax	ExpectedStDev
GBMS304-6	Au	ppm	4.58	4.2	4.96	0.19
GBMS911-1	Ag	ppm	11.9	9.9	13.9	1
GBMS911-1	Ag	ppm	11.9	9.7	14.1	1.1
GBMS911-1	Au	ppm	1.04	0.82	1.26	0.11
GBMS911-1	Cu	ppm	10028	9212	10844	408
GBMS911-4	Ag	ppm	17.9	15.1	20.7	1.4
GBMS911-4	Au	ppm	6.78	6.24	7.32	0.27
GBMS911-4	Cu	ppm	900	784	1016	58
ST05	Au	ppm	2.36	2.14	2.58	0.11
ST10	Au	ppm	3.37	3.03	3.71	0.17
ST17	Au	ppm	0.76	0.7	0.82	0.03
ST23	Au	ppm	0.087	0.047	0.127	0.02

11.3.4 Duplicate Samples

Field duplicates were inserted by DPMK in the SHA and UG datasets. Results of these along with laboratory duplicate and repeat analyses and pulp split analyses were reviewed.

Gold field duplicates appear to exhibit a bias to the original samples. However only a small population of duplicates have been analysed and no definitive conclusion can be made at this stage. It must also be noted that there is a large nugget effect in this mineralised body and that the diamond core field duplicate programme is relatively new so there is insufficient data to determine what the correlation between the original and field duplicates should ideally between.

The laboratory duplicates and repeats exhibit acceptable correlations with no bias.

11.3.5 QA/QC Review and Conclusions

The results for blanks, standards and lab duplicates (repeats and splits) for gold, copper and zinc for the UG, SHA and UGC datasets have been reviewed and summarised below. All samples were analysed at the onsite SGS laboratory.

- CRMs and blanks performed acceptably.
- Field duplicates exhibited a significant bias to original for gold (and a lesser bias for Zn). Copper have an acceptable performance with no bias.
- Lab repeats have acceptable performance with no bias.

The total number of QA/QC samples submitted for the period of the QA/QC review is presented in Table 19.

Table 20. Assay QA/QC details (SHA and UG datasets)

QA/QC Category	Count of Samples	% Total Samples	Ratio (QA/QC : Orig samples)
Samples	50,492	79.68%	
Field Duplicates	174	0.27%	1:290
Lab Checks	1800	2.84%	1:28
Standards	10,901	17.20%	1:5
Total	63,367	100.00%	

Notes: Samples refers to original samples for assay
Field Duplicates: refers to field duplicate samples
Standards: CRMs and blanks (DPMK and Lab)
% Total Samples: Per cent of total samples (including DPMK and lab QA/QC samples) submitted
Ratio (QA/QC : Orig Samples): Ratio of QA/QC samples to original samples submitted

11.3.6 QA/QC Recommendations

No fatal flaws were noted in either the database review or the QA/QC review. Recommendations include:

- Issues noted in the previous MRE (January 2013) such as instances of apparent misidentified QA/QC samples and anomalous blank results appear to have been addressed in this review period. Ongoing monitoring is required.
- Review of field duplicates should continue in order to determine the correlation between original and duplicate results. A bias was noted in the gold field duplicates, which requires ongoing monitoring (only a small sample population was analysed).
- Currently, QA/QC is monitored on a batch by batch basis. QA/QC should be reviewed over longer period to assist with noting trends in the results (such as analytical drift).
- Umpire 'third party' assaying at an accredited laboratory is recommended as part of a comprehensive QA/QC procedure. QA/QC material should be included with the umpire samples.

CSA considers that the systems put in place by DPMK are sufficient to maximise the quality of drill hole samples and to assess the reliability, accuracy and precision of the assay results obtained.

Note: No DPMK QA/QC data was available for the underground channel samples (laboratory QA/QC material was reviewed) or the historical data and this QA/QC review applies only to the gold, copper and zinc results for the DPMK exploration.

11.4 Pre-DPMK Sampling

11.4.1 Diamond Drill Sampling

Underground diamond drill core sample intervals typically honoured vein or lithological contacts but were typically minimum 0.5 m, maximum 1.5 m lengths. Samples were typically taken from any area containing visible mineralisation (veins, disseminated sulphides) and extended sampling 4 m either side of such a mineralised zone. However, there appears to be variation in sampling procedures within the historic dataset during different periods of exploration.

Typically core diameters of 76 mm or less were sent for whole core analysis. Above 76 mm core was split and half the core was sent for assay whilst the remainder stored in core storage facility. All core storage facilities in Shahumyan have been destroyed and no useable core exists.

11.4.2 Underground Grade Control Sampling

Underground grade control sampling is classified as undifferentiated however it is essentially short, diamond drill, core holes.

- Sample intervals honoured vein or lithological contacts but were typically minimum 0.5 m, maximum 1.5 m lengths.
- Sampling procedures essentially followed the same as the diamond drill sampling.

Grade control samples were analysed using the same analytical methods as described for diamond samples in Section 11.4.1.

11.4.3 Underground Channel Sampling

All mineralised faces were sampled at 4 m intervals (every second face advance) with crosscuts routinely sampled at 1 m intervals. All sampling honoured geological boundaries with sub-samples.

- Samples were taken across a height of 10 cm to a depth of approximately 2 cm. Samples were taken to the limit of the underground development.
- Samples were collected using a hammer and chisel.
- Intervals were marked using charcoal/chalk.
- The sampler would stand on a sheet of fabric which acts to catch the sample interval from where it was placed in calico bags.
- Minimum sample weights were 4 kg to 7 kgs.

Channel samples were analysed using the same analytical methods as described for diamond samples in Section 11.4.1.

11.5 Pre-DPMK Sampling Analysis

Generally all samples were analysed for Cu, Au, Zn, Ag and Pb. Between 1972 and 1976, scientific research institutions from Moscow also conducted limited core assay for S, Fe, Se, Te, Cd, Ga and In. Typically samples were sent to various independent laboratories within Armenia and the greater Caucuses area for assay. A summary of assay methods is presented in Table 21.

Table 21. Summary of assay methods

Element	Period of work	Method of analysis
Cu	1962-1994	Iodometric titration
Pb & Zn	1962-1980	Polarographic method
Pb & Zn	1980-1994	Iodometric titration
Ag + Au	1962-1994	Fire Assay

11.6 Pre-DPMK Sample QA/QC

No historical QA/QC data is available for either the drill or channel data.

12 Data Verification

12.1 Sample Type Review

A large proportion of the MRE dataset is reliant on historical data as well as data obtained from a variety of sampling methods. These methods include:

- Underground Historical and DPMK Diamond Drilling (“UHD” and “UD”)
- Underground Historical and DPMK Channel Sampling (“UHC” and “UC”)
- Underground DPMK Grade Control Drilling (“UDGC”)
- Surface DPMK Diamond and RC Drilling (“SD” and “RC”)
- Surface Historical and DPMK trenching (“HST” and “ST”).

A statistical comparison was completed to assess compatibility and representivity of the different hole types for use in the MRE. The review compared grade trends for each hole type against each other and spatially using:

- Summary statistics
- Probability plots, illustrated in Figure 29, and histograms
- Swath plots
- And by undertaking Kriged and Nearest Neighbour estimates of larger veins using combinations of the different datasets.

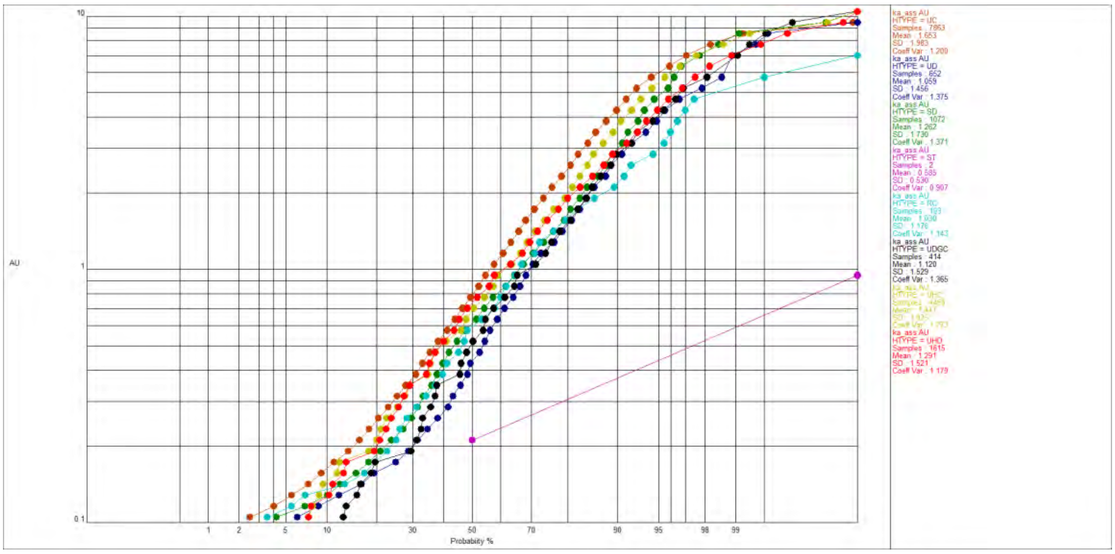


Figure 29. Probability plot for each HTYPE for all data within mineralisation wireframes for 0.1 g/t < Au < 10 g/t.

Based on the review work, it was difficult to reconcile the grade populations of the different sample/drill types. Grade distributions followed similar general trends however globally their grade averages are different, as illustrated in Figure. Also both Kriged and Nearest Neighbour estimates showed significant different results if channel data were or were not used (grades were sometimes >203% higher if channels alone were used).

Observations and documentation regarding drilling and sampling procedures at Kapan and historically, are that appropriate standards are typically met by all drill/sample types and that there is no obvious reason for bias. The review highlighted that the difference in grade distributions between the different drill/sample types is attributed to:

- UD and UHD as well as UC and UHC datasets have sampled different portions (i.e. grades) of the Shahumyan resources, during different periods of resource development, as illustrated in Figure 15 and Figure 16. There are no areas that allow for a 1:1 comparison.
- Drill data have lower global grades as they typically intersect large lengths waste.
- Attempts to compare grade distributions using data flagged within the Vein Model was complicated by poor geological logging to constrain mineralisation intercepts and the fact that a large proportion of high grade intercepts fall outside the vein model.
- The above point is further complicated by the fact that mineralisation is nuggetty and metal proportions vary within single veins preventing 1:1 comparisons for different datasets that have intersected the same vein.

Based on this, CSA is of the opinion that there are no obvious reasons for sampling bias.

Note: In the January 2013 CSA MRE (CSA,2013[2]); the UDGC data was removed from the MRE due to the irreconcilable grade distributions, the apparent bias was supported by poor underground methodology for channel sampling, observed during Principal Geologist Galen White's site visit in May 2013. Subsequent visits by CSA Principal Geologists Steve Rose and communications with DPMK have now confirmed that the sampling errors observed during the May 2013 visit was not standard DPMK procedure. During the May 2013 visit the Hilti drill tool had broken and conventional hammers were used, favouring sampling of less-competent, vein material. The Hilti Drill has been fixed since August, and additional tools are on order.

Based on the review work undertaken by CSA as well as observations made onsite made by CSA staff, it is believed that all drill and channel sampling data are sufficiently reliable to be included in the estimation of resources.

Historical Surface Trench Data ("HST") and DPMK Surface Trench Data ("ST") were excluded as these sampled only surface mineralisation which is immaterial to this MRE.

12.2 Geological Logging

CSA considers the onsite procedures employed to collect and capture geological observations are appropriate for this MRE; however the following observations are made regarding the data captured:

- When reviewing logged lithologies, it is difficult to establish a correlation between grade and rock type. This is exemplified by:
  - The prevalence of mineralised intercepts that fall outside of the vein model, which have not been classified with a rock code indicating either a vein or a mineralised structure.
  - The dominance of lithologies other than vein that are flagged within the Vein Model.
- Logging of core to 1 m intervals should be replaced with logging to intervals of geology and alteration, and not strictly to 1 m intervals.
- CSA believes that lithological logging should be better placed to target mineralisation and confirm and support continuity of grades between drill holes, which is currently informed predominantly by grade outlines.

Table 22. Top ten lithologies flagged within the Vein Model

Lithology	Count	% of total samples
Andesite	5201	82%
Lapilli Tuff	210	3%
Hydrothermal Breccia	153	2%
Vein	122	2%
Volcanic Breccia	109	2%
Sedimentary Clay	105	2%
Dacite	71	1%
Conglomerate	68	1%
Pyroclastic Breccia	54	1%
Quartz Vein	42	1%

12.3 Bulk Density

Approx. 49,000 density values were provided. Of these, 43 samples had a density bottom cut to 2 g/cm^3 and 26 samples had densities topcut to 4.25 g/cm^3. Of these cut samples; 90% were logged as VAN (volcanic andesite). These cuts resulted in <0.1% difference in the dataset's average SG.

Density data was flagged by the vein model and desurveyed with the lithology and assay data to review the density relationship w.r.t. to geology, grade and location. Review work replicated the results of the previous MRE whereby there appears to be poor geological control regarding the density data. In summary:

- The global dataset SG was 2.67 g/cm3, of this 76% of the data was logged as VAN with an average SG of 2.67 g/cm3.
- 70% of all densities >3.9 g/cm3 were logged as VAN.

- 80% of the data flagged within the vein model was logged as VAN, with an SG of 2.71 g/cm3.
- There is no defined relationship between lithology or location and densities; although it understood that vein material is denser.
- Some veins are denser than others, however there is insufficient density data flagged by each individual modelled vein to classify them separately.
- Veins at Kapan are typically poly-metallic with variable associated densities.
- There is a clear relationship between density and grade, illustrated in Figure 41 supporting the use of a regression to estimate density from grade.

12.4 Drill Recovery

12.4.1 Diamond Core Recoveries (Surface and Underground)

The large majority of DPMK drilled surface and underground diamond drill holes have recovery data (92% of the holes available to the MRE) and this data show no issues with regards to recoveries with recoveries in excess of 99% typical, as presented in Table 23.

Table 23. Diamond Drill (DD) holes in the MRE with recovery data and average recoveries

Hole Type	Average Recovery	MRE Holes with Recoveries	Total MRE holes
Surface DPMK DD	99	553	598
Underground DPMK DD	99.9	85	92
Underground Pre-DPMK DD	n/a	0	1482

12.4.2 RC Recoveries (Surface)

Sample weights were available for 142 RC holes. Poorer recoveries were typical at shallower depths (above 35 m) due to intersecting weathered/oxidised units, as illustrated in Figure 30. A visual review of the data showed that poor recoveries were predominantly above any mineralised units.

However there remains some spread in the recoveries for intervals below 35 m depth. Target sample weights are between 32 to 38 kg based on a 5.25” bit.

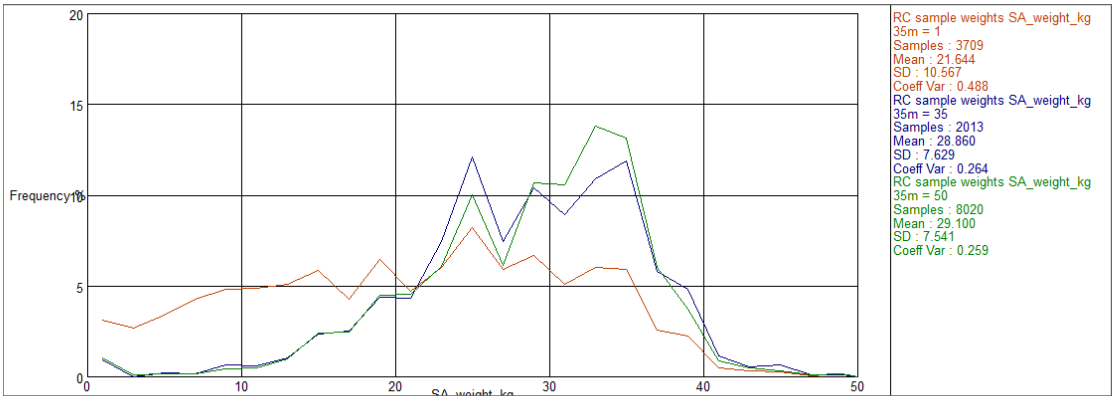


Figure 30. RC recovery weights (kg) for samples above 35 m (blue), above 50 m (green) and below (red).

12.5 QA/QC Data Verification and Validation

The Kapan QA/QC data was loaded into an SQL database and reviewed in QAQCR, a software package designed to monitor and analyse quality control.

Control charts for blanks and certified reference materials ("CRMs") as well as scatter plots of field duplicates and pulp splits were plotted. Assay values were plotted as well as expected values and their upper and lower ranges (twice the standard deviation from the expected value). An example of the results of a gold CRM (ST05) is illustrated in Figure 31, where no failures occurred.

No fatal flaws were noted in the QA/QC review which indicates that the QA/QC implemented at Kapan is sufficient to maximise the quality of drill hole samples and to assess the reliability, accuracy and precision of the assay results obtained.

Note: No DPMK QA/QC data were available for the underground channel samples (laboratory did insert QA/QC material) or the historical data and the QA/QC review applies only to the DPMK exploration (UG and SHA datasets) for gold, copper and zinc. The results of this review are reported in Section 11.3.

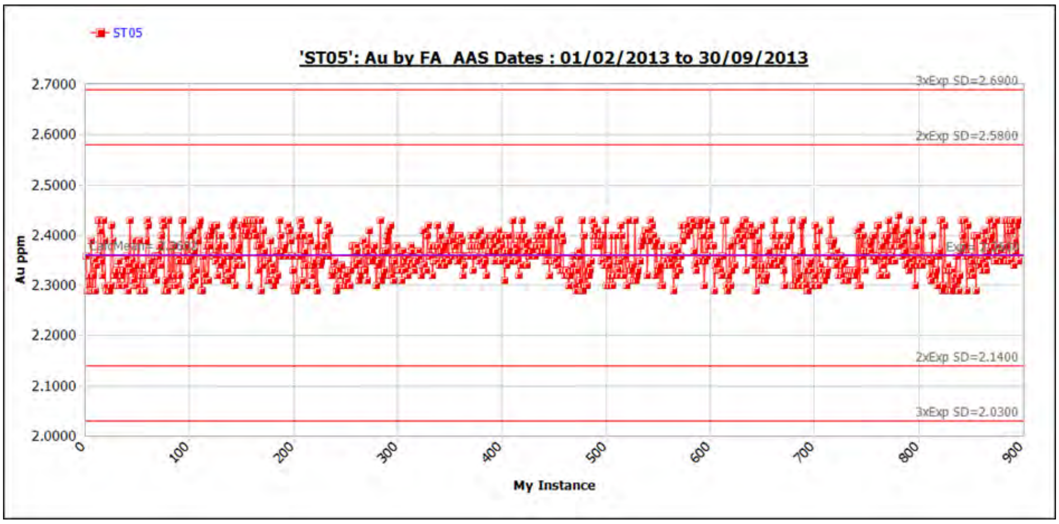


Figure 31. Gold CRM ST05 showing results plotting around the expected value.

12.6 Database Verification and Validation

12.6.1 Introduction

DPMK provided database exports in CSV file format. The cut-off date used for this resource update was the 5 December 2013. Data for 15,949 drill holes and channels were loaded into DataShed using SQL and validated. The data review relates to the datasets presented in Table 24, provided by DPMK:

Table 24. Datasets Included in the Resource Estimation

Dataset	Description
UG	DPMK Underground Drilling
SHA	DPMK Surface Drilling
UGC	Underground Channel Sampling
UGHC	Underground Historical Channel Sampling
UGHR	Underground Historical Drilling
UGST	Underground Grade Control

Table 25. Database Export files provided by DPMK (CSV formats)

File Name	Comments
DPMKa_Assay_20131004.csv	Kapan Resource Data
DPMKa_Assay_BESTEL_20131004.csv	Kapan Resource Data
DPMKa_Assay_Methods_Ranking_20131004.csv	Kapan Resource Data
DPMKa_Assay_Translation_20131004.csv	Kapan Resource Data
DPMKa_BD_20131004.csv	Kapan Resource Data
DPMKa_Collar_20131004.csv	Kapan Resource Data
DPMKa_Geology_20131004.csv	Kapan Resource Data
DPMKa_Labstandards_20131004.csv	Kapan QA/QC Data
DPMKa_Magsusc_20131004.csv	Kapan Resource Data
DPMKa_Orientation_20131004.csv	Kapan Resource Data
DPMKa_Standard_Expected_Values_20131004.csv	Kapan QA/QC Data
DPMKa_Standards_20131004.csv	Kapan QA/QC Data
DPMKa_Structure_20131004.csv	Kapan Resource Data
DPMKa_Validation_Codes_20131004.csv	Kapan Resource Data
DPMKa_DHSurvey_20131004_ver2.csv	Kapan Resource Data
DPMKa_Geotech_20131004_ver2.csv	Kapan Resource Data
DPMKa_Pairs_20131004.csv	Kapan QA/QC Data
DPMKa_Geology_20131004_2.csv	Additional Kapan Resource Data
DPMKa db changes.xlsx	Corrected Kapan Resource Data
DPMKa_DHSurvey_20131004.csv	Superseded by DPMKa_DHSurvey_20131004_ver2.csv
DPMKa_Geotech_20131004.csv	Superseded by DPMKa_Geotech_20131004_ver2.csv

The following sub-sections describe the data in the database and the validation steps undertaken by the database manager to verify the data used in the technical report. These include:

- Data verification procedures undertaken when importing the data.
- Summary of data in the database.
- Opinions of the database manager regarding the reliability of the data for the purposes of the technical report.

12.6.2 Data Loading and Validation Summary

The list below includes validations and checks carried out in the data validation process for each database table:

- Collar table: Incorrect co-ordinates (not within known range), duplicate holes;

- Survey table: Duplicate entries, survey intervals past the specified maximum depth in the collar table, overlapping intervals, abnormal dips and azimuths;
- Geotech table: Core recoveries greater than 110% or less than 0% and RQDs greater than 100% or less than 0%, overlapping intervals, missing collar data, negative widths, geotech results past the specified maximum depth in the collar table;
- Geology, Sample and Assay tables: Duplicate entries, lithological intervals past the specified maximum depth in the collar table, overlapping intervals, negative widths, missing collar data, missing intervals, correct logging codes, duplicated sample ID's, missing samples (assay results received, but no samples in database), missing analyses (incomplete or missing assay results).
- QA/QC material: A QA/QC (Quality Assurance, Quality Control) report is generated in which results of the standards (CRMs), blanks and duplicates are reviewed (includes client QA/QC material and lab checks where applicable).

The data in the UG and SHA datasets are comprehensive, of a high standard and no significant issues were noted. Less data were available for the historical and channel datasets; no geology, metadata or QA/QC data were provided for these datasets.

12.6.3 Location

There are 15,949 drill holes in the collar file and all of them had co-ordinate data.

<u>*UTM Grids*</u>

The UTM grids, presented in Table 26 (with grid extents) are referenced in the data:

Table 26. UTM grids (with grid extents) referenced in database

DataSet	Orig_Grid_ID	Min_East	Max_East	Min_North	Max_North	Min_RL	Max_RL	Hole Count
SHA	WGS84_38N	8622376.80	8624554.29	4342674.48	4344926.68	763.73	1091.153	739
UG	WGS84_38N	8623018.00	8623976.28	4342878.88	4344608.03	701.87	771.5	176
UGC	WGS84_38N	8622862.86	8624210.00	4342874.67	4345008.06	617.22	873.08	6853
UGHC	WGS84_38N	8622858.94	8624487.92	4342696.63	4345037.09	510.50	873.33	5820
UGHR	WGS84_38N	8619756.97	8625983.06	4342175.70	4347594.70	510.43	1137.83	1501
UGST	WGS84_38N	8622939.76	8624354.51	4342709.17	4344939.16	510.50	871.68	860

- No holes have missing UTM co-ordinates or missing RLs.

<u>*Collar Survey Methods*</u>

Co-ordinates are captured at various stages using different methodology, ranked accordingly and those with the highest (best) ranking are captured in the 'Best' field in the database. These co-ordinates were used in the mineral resource estimation.

Highest to lowest ranked methods are as follows:

DGPS->Total station->Digitizing->Transformation->Planning

Associated event depths are also reviewed to identify any obvious concerns, such as depths beyond the maximum depth of the drill hole. There are no issues with the data in the collar table.

12.6.4 Down Hole Surveys

All drill holes have down hole survey records (51,355 records) and there are no validity issues with the intervals or depths. Dips should be between -90 and 90 degrees. All surface drill holes should have a negative dip. There are 21 UG drill holes containing portions with positive dips, usually towards the end of flat holes, which is not an issue as these are underground holes. As expected, there are no holes with positive dips in the SHA dataset. Azimuths should be between 0 and 360 degrees. There were no issues with Azimuths.

12.6.5 Sampling

There are 351,323 samples in the assay table, Table 28 below lists the number of samples and holes missing samples. Six holes in the UG dataset and ten in the SHA dataset have no sampling data as either the logging and sampling of these holes was incomplete when the database was exported or they are geotechnical holes.

Non-unique sample numbers have been used and are detailed below. No issues with interval integrity such as overlapping intervals, from depths less than to depths, and intervals greater than the specified maximum hole depth have been noted. Dummy samples (NS) were created for intervals that were not sampled but are not included in the counts above.

There are 6,131 samples with no associated assays.

Sample Recovery

Sample recovery data has been provided for the SHA, UG and UGHR datasets. 50,261 samples (including channel samples) do not have sample recovery data, as presented in Table 27.

Table 27. Samples with and without Recovery Data

DataSet	Sample Type	Samples - Rec	Samples - No Rec
SHA	DD	177,929	15
SHA	RC	13,671	0
UG	DD	56,486	288
UGC	CH	0	23,820
UGHC	CH	0	20,367
UGHR	DD	52,976	179
UGST	DD	0	5,592
TOTAL		**301,062**	**50,261**

12.6.6 Geological Data

There are 248,350 records in the lithology table for the UG and SHA holes and there is no geological data for any of the other datasets. The table below lists the number of geological records in the database. The UG holes missing geological data (two) were unfinished at the data export cut-off date.

There are no overlapping intervals, from depths less than to depths, or intervals greater than the specified maximum hole depth in the geological tables.

Geological Codes

Geology is described by the use of codes. Descriptions were provided in the CSV file 'DPMKa_Validation_Codes_20131004.csv'.

12.6.7 Bulk Density Data

There are 54,048 bulk density samples in the UG and SHA datasets of which 4,795 did not have bulk density or SG assay results as of the cut-off date.

12.6.8 Data Summary

Records in the Kapan database are presented in Table 28.

Table 28. Summary of Records per Hole Type in the Database

DataSet	Hole count	Type	Sample count	Holes (No Samples)	Assay count	Samples (No Assays)	Lith record count	Holes (No Lith)	DH Survey count	Holes (No DH Survey)
SHA	598	DD	177,944	8	177,849	95	179,461	0	13,489	0
SHA	141	RC	13,671	2	12,992	679	13,973	0	1,312	0
UG	176	DD	56,774	6	52,026	4748	54,916	2	2,201	0
UGC	6,853	CH	23,820	0	23,799	21	0	6,853	11,830	0
UGHC	5,820	CH	20,367	0	20,319	48	0	5,820	14,097	0
UGHR	1,501	DD	53,155	0	52,626	529	0	1,501	2,660	0
UGST	860	DD	5,592	0	5,581	11	0	860	5,766	0
Total	**15,949**		**351,323**	**16**	**345,192**	**6,131**	**248,350**	**15,036**	**51,355**	**0**

12.7 Site Visit

Mr Galen White of CSA Global (UK) made a site visit to the Kapan project in Armenia in February 2013 (CSA,2013[1]) and has been in regular communication with DPMK site staff since July 2013 to the present time. A summary of site visit observations are presented below and are tabulated in Table 29.

Malcolm Titley (Principal Consultant CSA UK) conducted a site visit for a period of three days between 19 and 22 August 2013. Site MRE procedures were reviewed as part of this site visit.

CSA Principal Geologist, Mr Steve Rose has also visited Kapan for the purposes of reviewing mine geology aspects, completing a mine geology grade control gap analysis and advising DPMK on areas of improvement.

The following conclusions were made during the visit, and remain current:

- The data collection procedures in place during resource development drilling, mapping and channel sampling activities are appropriate for use in resource estimation work.
- Metadata collected from the various works are detailed and appropriate for investigations of the different data types over time.
- QA/QC data from the various works are routinely reviewed and actions taken where results are out of tolerance. However the site procedures for formal monthly QA/QC report generation needs to be clarified and set-up to allow tracking of results over time.
- Software in use on site is appropriate and consistent across departments. The use of acQuire for data capture and storage is robust and a very good data model is in place to capture metadata, geological, geotechnical, density, and analytical data which is managed on site by a dedicated database manager, reporting to an overall data manager for the group, situated in Chelopech.

- Core yard procedures and protocols are in place to ensure core yard functions are performed with adequate controls in place, and that information collected is reliable and sampling error is minimized. Checks completed on core and sampling produced no issues.
- A brief inspection of the SGS Laboratory facility in Kapan suggests that this facility is under good management and standard SGS systems are in place. The laboratory is internationally accredited.

Table 29. Summary observations from Site Visit by Mr White and confidence levels of key categorisation criteria (CSA,2013[1]).

Kapan Project (Underground Resource Estimation Component) Confidence Levels of Key Categorisation Criteria		
Items	**Discussion**	**Confidence**
Drilling and Channel Techniques	DPM Diamond and RC drilling was undertaken to an appropriate standard.	High
	Historical diamond drilling and channel sampling techniques, practises and control cannot be verified. Only hard copy plans/sections remain. No core remains.	Low
Logging	DPM Geological logging is completed using appropriate practises and standard nomenclature. Detailed metadata collected. Logging is only summary in part.	Moderate
	Historical Geological logging in unknown, and cannot be verified.	Low
Drill Sample Recovery	DPM: Acceptable recoveries determined for the majority of the drilling inspected at the core facility. A review of the database records suggests adequate core recoveries.	High
Sub-sampling Techniques & Sample Preparation	DPM diamond and RC drilling was undertaken to appropriate standards	High
Quality of Assay Data	Extensive quality control data available for DPM data only and has been assessed. The majority of quality control data is acceptable but there is limited QA/QC available UG channels.	Moderate
Verification of Sampling and Assaying	Umpire assaying completed which indicates robust analytical data.	Moderate
Location of Sampling Points	All DPM drill hole collars and have been surveyed and most drill holes have been down hole surveyed.	High
	Historical drilling positions are uncertain, only taken from hardcopy data.	Low
Data Density and Distribution	The current data spacing (80 x 80, 40 x 40, 20 x 20) is considered appropriate for resource evaluation. Underground fans provide variable piercing, but representative intersections.	Mod/High
Database Integrity	acQuire database model is robust, and of an appropriate standard. A wealth of metadata and geological, geotechnical, density and other data is being collected. Database control is managed on site and from group DM administrator in Chelopech.	High
	Historical database not readily verifiable.	Low
Geological Interpretation	The interpreted lithological and weathering boundaries based on the DPM data are considered robust and of high confidence. Mineralisation boundaries are on AuEq grade constraints and no domaining of HG zones.	Moderate
Mining Factors or Assumptions	Current in-house model uses 3 x 3 x 3 blocks which apparently honour the mining SMU. To be reviewed with input from the client.	Moderate

Tonnage Factors	In-house estimate uses blanket BD assignation. An improved 3D geological model should allow improvements to be made to the assignation of density to the model,	Moderate

13 Mineral Processing and Metallurgical Testing

13.1 Historic Test Work

A summary of metallurgical testwork that was undertaken in 1975 to investigate rare metals at the Shahumyan deposit (Dubchak,2009) took place between 1975 and 1976 and covered the deportment of minor metals into three concentrates produced by sequential flotation. The studies were conducted for the 1976 Reserves report in 3 specialized metallurgical research institutions:

- Armenian State Institute for Base Metals Projects ("ArmHYPROTSVETMET") where the composition of mined material was studied
- Laboratory of the North Caucasian Mining and Metallurgical Institute, where recoveries for mined material and concentrate were studied in a laboratory conditions
- Soviet Scientific Research Institute of Base Metals ("VNIITSVETMET"), where a semi-industrial test was conducted on a 1300 t combined sample collected from 12 bulk samples from 4 levels.

It is noteworthy that the average grades for the main commodities in the metallurgical sample were significantly higher than those for regular Shahumyan mineralised material.

Possible recovery technologies for those elements were tested only briefly in the laboratory of ArmHYPROTSVETMET. The tests returned, that:

- The rare metals could satisfactory be extracted by cyanidation, and
- The recovery of main commodities from combined Cu- Pb concentrate improved after a cyanide treatment by 9.4%: Cu- up to 78%, Pb- up to 73.2%.

However, loss of Au increased by 29% with cyanidation, which made this option not expedient.

The best recovery for the main commodities, according to the research, was achieved by producing of 4 types of the concentrate. The grades and recoveries of the main commodities and rare metals into those products are presented in Table 30.

Table 30. Rare Metals Recoveries into Metallurgical Products

Product	Recovery %	Au	Cu	Zn	Pb	Ag	Cd	Se	Te	Ga	In
	Grade, g/t- for Au, Ag and rare metals, %- for the base metals										
Mineralised Material	100	6.5	0.97	6.64	1.17	149	400	14	10	13	9
Cu concentrate	2.29	59.7	30.5	2.0	6.8	1235	420	32	920	4	95
Pb concentrate	2.28	109.0	6.5	8.3	36.0	3270	110	190	2320	12	43
Zn concentrate	11.4	16.0	0.6	52.5	0.51	281	3200	3	280	66	36
Py concentrate	10.0	5.1	0.27	1.8	0.49	74.5	40	34	10	3	4
Tails	74.04	0.44	0.04	0.34	0.12	10	50	4	10	6	2
Recovery into concentrate, %											
Cu concentrate	2.29	21.0	72.0	0.65	13.3	19.0	2.4	5.3	19	0.7	24.3
Pb concentrate	2.28	38.2	15.1	2.85	70.0	50.0	0.6	31.4	48.1	2.1	10.9
Zn concentrate	11.4	28.0	7.0	90	5.0	21.5	90.5	18.6	29	57.6	46.0
Py concentrate	10.0	7.8	2.8	2.7	4.2	4.5	1.0	24.1	4.8	2.3	4.1
Tails	74.04	5.0	3.1	3.8	7.5	5.0	5.5	20.6	9.1	37.7	14.7
Metallurgical recovery from concentrate into market commodity product, %											
Cu concentrate							-	73.26	65.7	-	-
Pb concentrate							99.66	-	-	-	53.0
Zn concentrate							89.66	17.01	-	-	50.3
Py concentrate							-	49.08	-	-	-
Through recovery from mined material into market commodity product, %											
Cu concentrate							-	3.9	12.5	-	-
Pb concentrate							0.59	-	-	-	5.8
Zn concentrate							81.1	3.2	-	-	23.1
Py concentrate							-	11.8	-	-	-
FINAL THROUGH RECOVERY							**81.69**	**18.9**	**12.5**	**0**	**28.9**

However, since the 4 concentrate types have never been produced on the commercial scale and the original recommendations have never been implemented in practice:

- The figures in the table remain as projections
- The issue of rare metals recovery into concentrate was put aside given the practical implications of producing concentrate using the Shahumyan mill and its treatment in different than proposed facilities.

In 1976 the GKZ of USSR recommended using the 4 concentrate scheme on Shahumyan as the main. GKZ agreed accepting the existing Kapan mill technology for Shahumyan mineralised material as a temporary solution, stating that a new line shall be commenced and additional metallurgical studies should take place. That temporary solution has been in practice of the mine for over 30 years.

13.2 Camborne School of Mines Associates (“CSMA”), 2003

Since the commencement of processing mineralised material from Shahumyan, the operation has produced two separate concentrates – a Copper concentrate containing significant gold and silver credits, and a Zinc concentrate, also with gold and silver credits. In 2003, a test program was initiated and included some basic characterisation of various plant and underground samples. These culminated in several 'locked cycle' flotation tests using the then current plant conditions – one of these is presented in Table 31.

Table 31. Metallurgical Prediction of Performance – Lock Cycle Test #1

Product	Weight %	Assay %					Distribution %				
		Au (g/t)	Ag (g/t)	Cu	Zn	Pb	Au	Ag	Cu	Zn	Pb
Cu Conc.	0.86	276	4462.6	19.3	9.9	16.7	76.7	70.2	83	6	84.4
Zn Conc.	1.85	22.8	403.2	0.5	63.6	0.39	13.5	13.6	4.8	82.1	4.2
Tails	97.29	0.31	9.16	0.03	0.18	0.02	9.7	16.2	12.1	11.9	11.4
Feed	100	3.11	54.94	0.2	1.43	0.17	100	100	100	100	100

- The 'locked cycle' tests demonstrated the recovery potential – from typical sample head grades (0.23% Cu, 0.19% Pb and 1.61% Zn), copper recoveries of 83-87% to a concentrate grade of 19-23%, and Zinc recoveries between 80-87% at a concentrate grade of in excess of 60% were shown to be achievable.
- Average Plant performance at the time is outlined in Table 32 to Table 35. At the time of testing, typical performance for the two concentrates were - 65% recovery of Cu, Au and Ag to a 15%Cu grade copper concentrate, and 65% recovery of zinc to 55%Zn grade zinc concentrate.
- Gold recoveries to copper concentrate ranged from 70-75% with approximately 15% of the gold reporting to the zinc concentrate.

13.3 Dundee Precious Metals, 2006

A significant plant survey and underground sampling program was completed by DPM prior to the acquisition of Deno Gold in 2006. SGS Minerals (Lakefield, Canada) undertook the assay checks of the underground samples, together with grinding and flotation testwork on the plant survey samples, and the individual and composite samples from the underground samples (Deno Gold,2006; Sloan,2006).

In total, 41 mine channel samples, and a set of survey samples were shipped to SGS Lakefield in May, 2006. The approach taken in the study involved:

- A full series of plant samples were taken on successive days covering the grinding and flotation circuit products to confirm current actual section performance being achieved.
- Grindability tests, including Bond Rod and Ball Mill Index Tests ("Bond WI"), SAG Power Index Tests ("SPI"), of the plant and selected channel samples.
- Conducting two types of laboratory flotation tests (MinnovEX Flotation Tests ("MFT")) to determine comparative rougher flotation kinetic parameters using a reagent suite developed through two preliminary rougher scooping tests.

- Conducting an SGS proprietary flotation simulation study (FLEET1) on the results of the laboratory flotation program to estimate the impact of mined material types, feed grade, mass pull and pyrite rejection upon grade-recovery at standard grind.

Conclusions drawn from both phases of work are summarized:

- Both reports at this time indicated there was significant potential for improvements in terms of flotation performance at the Kapan.
- The mined material is generally 'soft' by most hardness measures with a Bond Ball Mill Work Index of between 10-14 kWh/t and SPI values ranging between 14-45 minutes.
- Analysis of the plant survey samples of the 24 and 25 of May, demonstrated insufficient liberation of the target minerals in the flotation feed with the resulting poor flotation performance (particularly for the zinc circuit).
- Improved flotation performance in the Kapan concentrator were likely if the primary grind size was reduced to increase liberation, and improvements in the quality of reagents used are made:

Recommendations to come out of this work were:

- Produce a finer primary grind (< P_{80} of 80 microns), by improving cyclone classification.
- Continue laboratory work on various reagent combinations, and follow up with plant trials on the most promising.
- FLEET simulation results are based upon MFT roughers only. The cleaner and regrind circuit at Kapan should be benchmarked and locked cycled tests conducted to ascertain the effect of regrind and reagent addition upon cleaner kinetics.

13.4 Historical Plant Operations Performance

The current plant treated material from the underground Centralni operation, commencing in 1953 when some 300,000 tonnes of material grading 1.72% Copper were processed (Deno Gold,2006). This ramped up to over 1 Mt (supplemented by 35 – 45% of the feed being produced from an open pit) by 1972 which continued at this rate until 1985 when the open pit ceased production. The underground operation ended its continuous phase of operation in 1997 having produced a combined mill feed of 29.5 Mt and some 2.2 Mt of copper concentrate (grading between 18 and 20% Cu). The material was noticeably extremely 'clean' (Chalcopyrite being the principal Copper mineral), and metallurgical recovery was generally above 90%.

The rate of mineralised material processed from the Shahumyan mine processed since 2003 is presented in Table 32 to Table 35 (DPMK, 2014). It should be noted that the in the years before 2008 the plant was also treating remnant material from the Centralni underground mine before it was finally closed, and also various quantities of material (highly oxidised) stockpiled around the mining licence from the earlier exploration adits and mine excavation activities since the 1930s. 100% of underground Shahumyan mineralised material commenced in 2008, before operations were suspended for 2 quarters, after the collapse of metal prices during the world financial crisis. Operations recommenced in April 2009.

Table 32. Historical Treatment of Shahumyan Mineralised Material at Dundee Precious Metals Kapan.

Material Processed		Head Grades Treated			
Year	**Tonnes**	**Au, g/t**	**Ag, g/t**	**%Cu**	**% Zn**
2003	251,418	1.65	26.0	0.25	1.24
2004	310,966	2.07	29.5	0.26	1.05
2005	256,363	1.42	27.9	0.24	1.47
2006	211,982	1.37	31.6	0.26	1.54
2007	263,469	1.46	34.7	0.23	1.44
2008	239,229	1.91	46.7	0.34	2.07
2009	218,235	2.36	47.0	0.35	2.08
2010	428,865	2.33	42.1	0.34	2.22
2011	521,769	1.77	33.8	0.27	1.81
2012	509,419	1.56	32.2	0.25	1.67
2013	465,894	1.85	34.3	0.27	1.68

Table 33. Historical Treatment of Shahumyan Mineralised Material (Cont.), Copper Concentrate produced.

	Copper Concentrate		Metal Recovery, %		
Year	**Tonnes**	**Grade, %Cu**	**Au**	**Ag**	**Cu**
2003	3,337	14.0	54.8	58.3	74.6
2004	4,960	12.4	68.0	68.8	76.4
2005	3,763	13.5	58.3	61.2	82.0
2006	2,402	16.3	54.0	48.8	72.1
2007	2,276	19.9	58.1	58.6	75.9
2008	3,322	20.5	68.9	68.8	84.5
2009	3,155	22.0	74.2	73.7	90.6
2010	6,396	20.5	72.9	74.1	90.9
2011	6,475	19.7	68.6	69.7	90.4
2012	5,508	20.2	68.0	68.8	87.9
2013	5,650	18.8	69.7	69.8	85.0

Table 34. Historical Treatment of Shahumyan Mineralised Material (cont.), Zinc Concentrate produced.

	Zinc Concentrate		Metal Recovery, %		
Year	Tonnes	Grade, %Zn	Au	Ag	Zn
2003	3,666	56.3	12.0	13.1	66.3
2004	3,486	53.3	7.5	8.0	56.9
2005	4,802	52.9	14.7	14.0	67.5
2006	3,897	52.6	16.0	14.2	62.5
2007	4,872	52.9	15.9	13.3	68.0
2008	7,451	53.7	14.2	13.6	81.0
2009	6,989	59.5	15.3	14.1	91.5
2010	14,361	60.3	17.9	16.7	90.9
2011	14,622	57.8	16.6	16.6	89.7
2012	11,347	61.7	17.5	16.4	82.5
2013	11,995	57.8	18.3	18.2	88.6

Table 35. Historical Gold and Silver Production in the Cu and Zn concentrates.

	Combined Recovery, %		Contained Ounces	
Year	Gold	Silver	Gold	Silver
2003	66.8	71.5	8,902	150,202
2004	75.6	76.8	15,654	226,440
2005	72.9	75.2	8,540	172,697
2006	70.0	63.0	6,537	135,855
2007	74.0	71.9	9,187	211,052
2008	83.1	82.4	12,210	296,033
2009	89.6	87.8	14,836	289,692
2010	90.8	90.8	29,215	527,362
2011	85.3	86.2	25,474	492,137
2012	85.5	85.1	21,843	449,081
2013	88.0	88.0	24,359	452,762

13.5 Mineralogical Studies

A series of investigations have been completed to identify the nature of the nature and mineral liberation contained in both the copper and zinc concentrates produced. SGS Lakefield (Canada) February 2011 overviewed both by “QEMSCAN”. More detailed investigations into the precious metal occurrences were completed on the Zinc Concentrate (SGS Lakefield, June 2011) and on the Copper concentrate in 2012. Both concentrates showed excellent liberation of each of the dominant minerals present, with the zinc concentrate exceptional in this regard.

In the copper concentrate, the dominant copper mineral is chalcopyrite (~60%, of which >50% is ‘free’ and liberated), with minor pyrite (~15%) , galena and sphalerite (~11% each). All other sulphides and gangue are < 1%.

In the zinc concentrate, sphalerite is dominant (~93%, and 95% ‘free’), minor pyrite (<4%), quartz (<2%) and all other minerals are <1%.

14 Mineral Resource Estimates

14.1 Mineral Resource Estimate Data

14.1.1 Sample Data

Data used in the preparation of the MRE was sourced from both surface and underground drilling and sampling completed over the project, the majority of which comprised underground channel sampling. The final data used in the estimation was taken from a database export provided by DPMK on 4 October 2013. Tables were imported from comma delimited text files into Datamine, and comprised the following:

- Collar Data
- Survey Data
- Assay Data
- Lithology Data
- Structural Data
- Density Data

Validation of the data consisted of software checks including the following:

- Missing coordinates or missing total depths
- Duplicate or overlapping intervals
- Duplicate holes
- Intervals continuing below the total depth specified for the drill hole
- Valid azimuths (0 to 360°) and dips (+90° to -90°).

Any errors flagged by the software checks were raised with and investigated by site personnel and the database amended, prior to continuing.

The original database export contained 15,949 drill holes. A polygon was used to restrict the data to that within the MRE area, illustrated in Figure 32, and trenches were removed, reducing the dataset to 15,932. The final data available for the MRE reported here is presented in Table 36 and is categorised by hole type.

Table 36. DPMK Data used in the current MRE

Drill hole Type	Hole Type Code	Number of Holes	Total Metres
Surface Reverse Circulation	RC	141	15,132
Surface Diamond	SD	597	181,309
Underground Channel	UC	6,853	22,816
Underground Diamond	UD	176	57,802
Underground Grade Control (undiff.)	UDGC	860	6,750
Underground Historical Channel	UHC	5,820	21,573
Underground and Surface Historical Diamond	UHD	1,485	270,952
Total		**15,932**	**576,335**

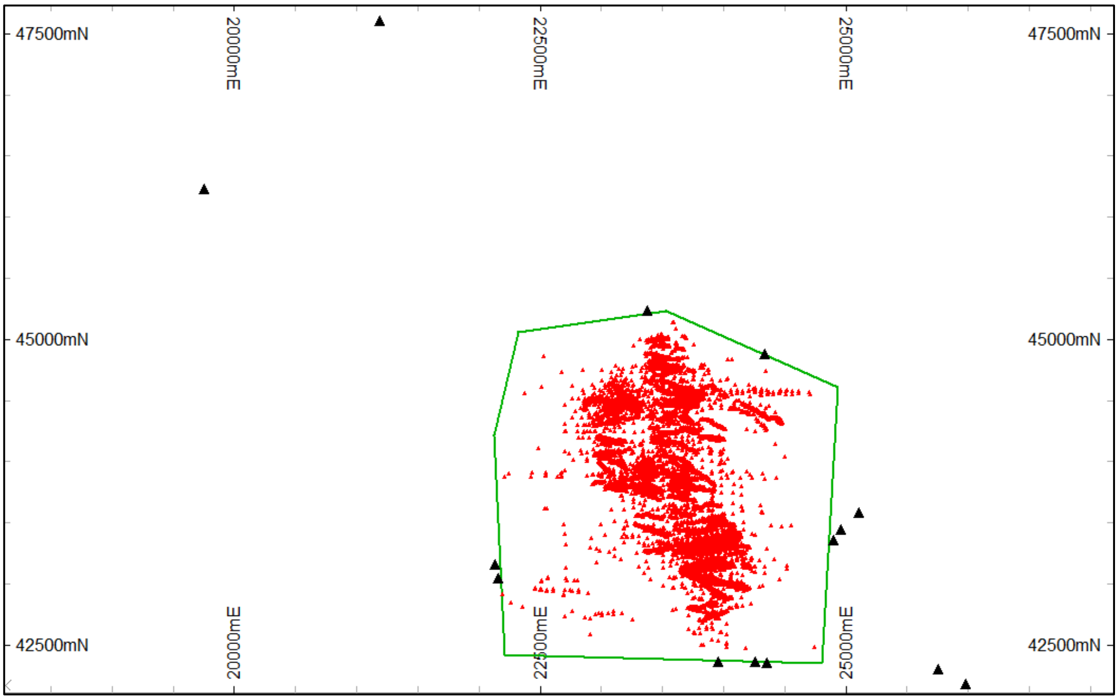


Figure 32. Plan view of the collars used in the MRE (red) and those excluded (black).

14.2 Geological Modelling

14.2.1 DPMK Vein Modelling

During 2012 and 2013 mineralisation wireframes were digitised by DPMK geologists. A nominal grade cut-off of 1.5 g/t AuEq was used in the interpretation to down hole lengths of

1 m. Underground mapping, sections and structural data taken from drill core were used to inform vein geometry and indicated that the dominant vein orientation has a strike approximating E-W, with a moderate to steep (60-90°) dip towards the south, as illustrated in Figure 33.

Leading up to this MRE update, a total of 158 individual vein solids, which included 24 updated or new veins from 2013, were supplied to CSA. Wireframe modelling by DPMK was completed in Surpac, Vulcan and GEMS and wireframe files were supplied to CSA in DXF format.

The vein wireframes supplied by DPMK were undiluted solids; these were based on assay intercepts that did not (necessarily) honour minimum mining width of 1.8 m horizontal thickness: the minimum width for Longhole Open Stoping method ("LHOS") employed at Kapan. CSA devised a process to broaden these interpretations to honour the minimum mining width of 1.8 m. A set of 'diluted' wireframes were created for this purpose.

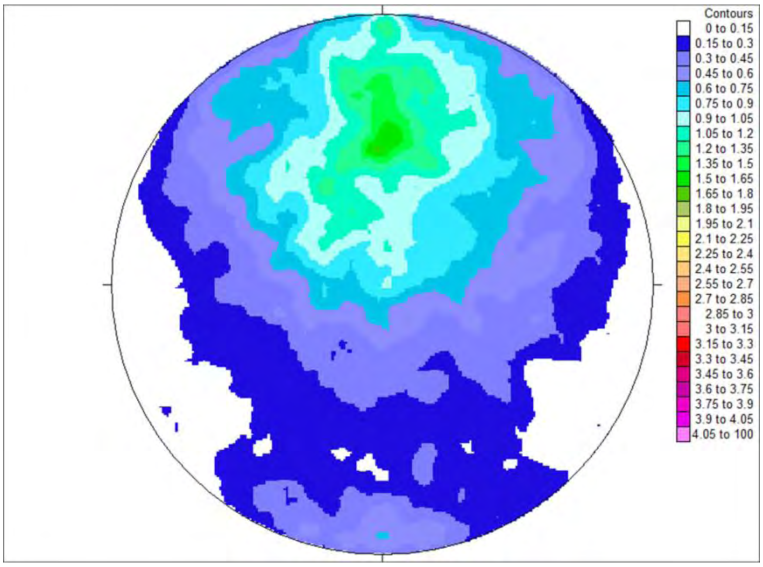


Figure 33. A stereographic projection (pole plot) of structural vein measurements taken at Shahumyan. Warmer colours indicate areas of clustered data.

14.2.1 Preparation of Data

Following validation, data was prepared so that it could be used in the modelling of diluted wireframes, for use in the MRE. Certain data needed to be treated carefully so as not to introduce bias.

- Missing data in other holes was assumed to be waste and assigned half detection values. If missing data was identified in the database as "Missing" (through core loss etc.) or if they were UDGC holes (where gaps within the drill string is used for the drive), the missing data was assumed to be absent and ignored during estimation.

- UDGC holes were historic holes drilled into either side of a drive with a Kempe drill. They were digitised off historic sections and now appear as a single hole in the database with a missing interval in the centre which corresponds to the drive itself and is often channel sampled. Such holes need to have the “missing interval” retained since setting it to waste is not appropriate, as illustrated in Figure.

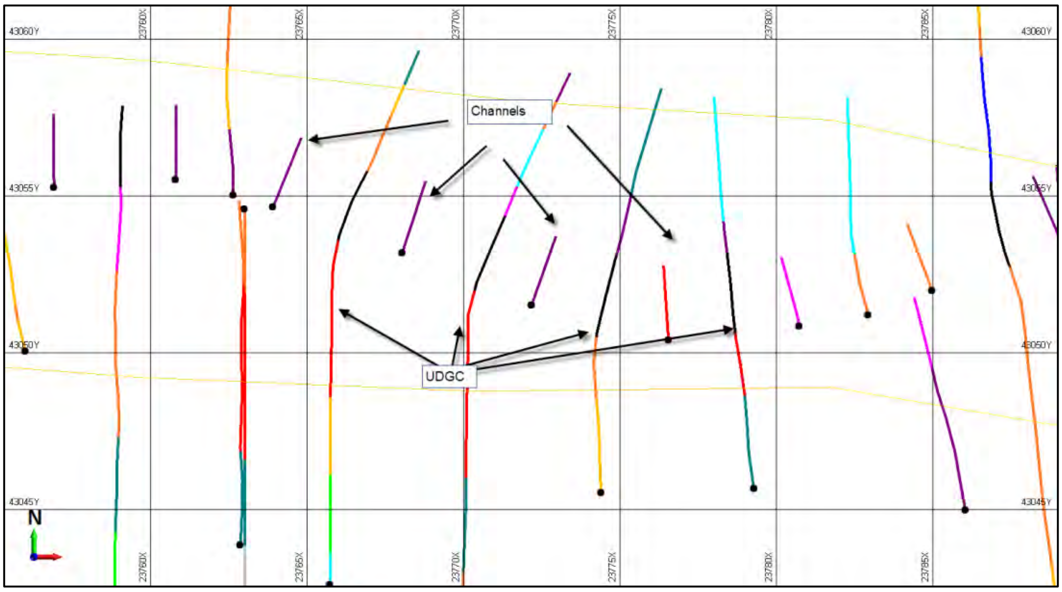


Figure 34. UDGC holes drilled into either side of a drive, with the “missing interval” coincident with the drive and channel samples (holes coloured by AuEq grade, missing intervals are black).

- In order to include the dilution necessary to honour the minimum mining widths, there were some channels and drill holes that needed dummy intervals inserted at the top and bottom of the channel / drill hole. Half detection limits were assigned to these dummy intervals.

14.2.2 CSA Diluted Vein Model

14.2.2.1 Preparation of Data

The following workflow was completed to generate the data file used to create the set of diluted wireframes used in the MRE.

- DPMK wireframes were converted to an XYZ points file containing dip and dip direction of the wireframe’s triangles, representing the localised geometry of modelled veins.

- Dip and dip direction were estimated into the desurveyed assay file using Nearest Neighbour interpolation method.

- Single length intercepts were created within each vein to obtain average dip and dip directions for the vein at the assay’s location.

- Assay data was then run through the True Thickness ("TRUETHK") and Length Compositing ("COMPSE") processes in Datamine software to identify mineralisation intercepts that met the required minimum mining width of 1.8 m and a minimum grade cut off of 1.5 g/t Au Eq.

  - TRUETHK calculates the true (horizontal) width of an intercept trigometrically; by incorporating the orientation of the drill hole and the orientation of the vein which flagged the intercept.

  - COMPSE preferentially selects optimal intercepts that meet or exceed the determined minimum length and grade cut off, moving up or down a hole trace, whilst iteratively seeking the highest grade.

- A 3 m x 3 m x 3 m block model flagged by DPMK wireframes was used to back flag sample data with vein number.

- The selected intercepts were filtered in SQL to identify maximum and minimum Northings of each intercept from which to create the northern and southern surfaces for the new diluted wireframes.

In addition to the automated process outlined above, some manual editing of data was required:

- To include intercepts at the terminations of the vein wireframes, where triangles were near-horizontal resulting in large minimum down hole lengths required by the TRUETHK process.

- To edit intervals that were incorrectly identified due to local anomalously oriented vein triangles, illustrated in Figure 35. These areas were reviewed on a case by case basis by using interactive grade compositing of grade data.

- To include portions of reasonable waste inside neighbouring vein structures in order to maintain demonstrated mineralisation continuity.

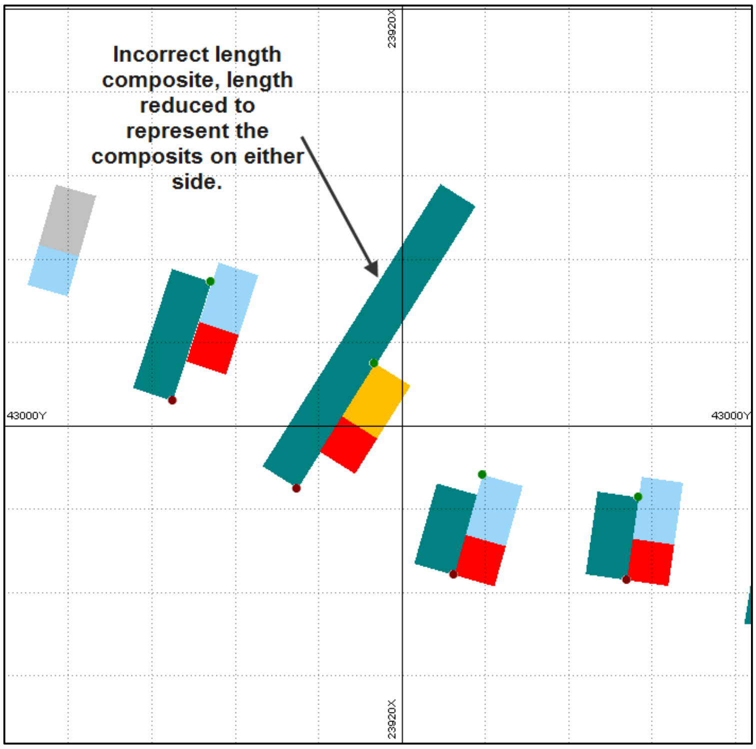


Figure 35. Example of erroneous Long COMPSE interval surrounded by shorted intervals, possibly due to sharp changes in wireframes leading to anomalous dip and dip directions of triangles.

14.2.2.2 Modelling of Diluted Wireframes

The north and south point's files file coded by vein number was loaded into Micromine and displayed as points alongside the original assay data and composited files.

North and south surfaces of each vein were created from the north point and south point files, filtered on vein code. For areas with more than 50 points Micromine Implicit modelling ("IM") was used to generate the two surfaces.

Inputs in to the IM point cloud modelling tool were derived iteratively, resolving a mesh size of 5 m, which best honoured the close spaced channel data. The use of implicit modelling resulted in vein orientation trends to be informed by real data points and produced realistically smoothed surfaces in a relatively short time frame, compared with conventional wireframing using strings. Implicit modelling was not possible on veins with fewer than 50 points; these wireframes were generated using point data and manual DTM creation North and south surfaces were created within bounding boxes, following which "cookie cut" wireframes were digitised to limit wireframes to within 30 m of supporting data, as illustrated in Figure 36. For wireframes digitised manually, surfaces were extended by 30 m horizontally and vertically by generating extension points.

Solids were validated for invalid connections, open sections, intersecting triangles and in 2D to ensure that they didn't pinch at the extremities to less than the minimum width. Where validation issues were observed wireframes were corrected manually.

In total 38 wireframes were generated using implicit modelling and 120 wireframes from DTM creation.

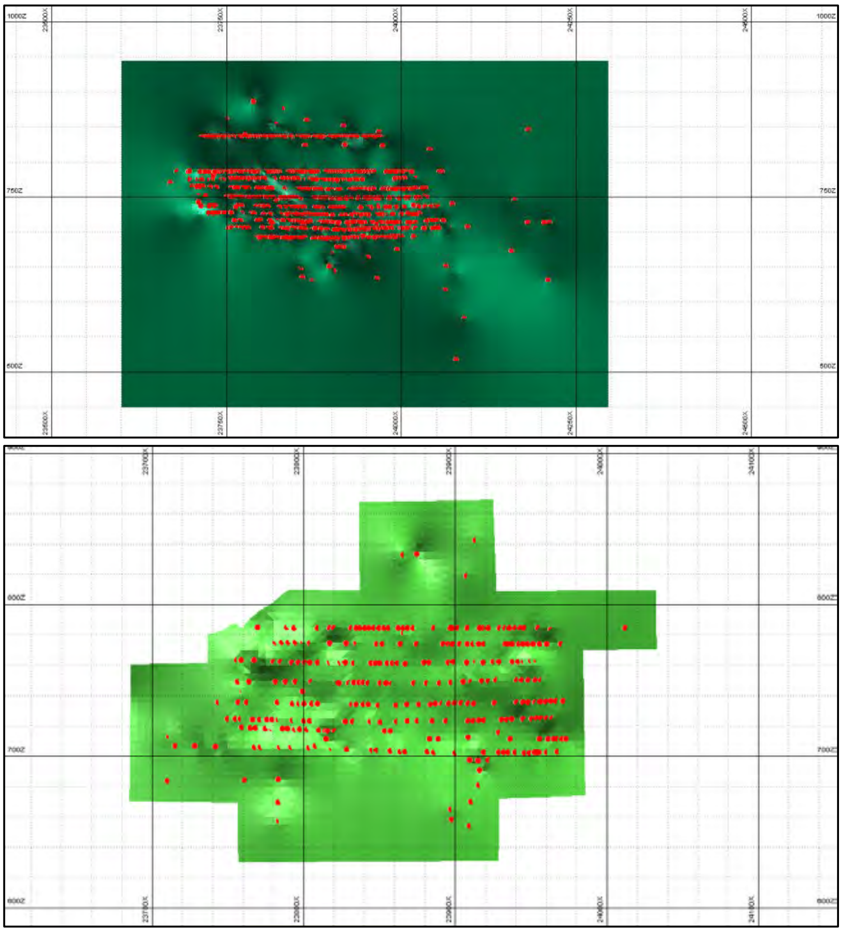


Figure 36. Top: Vein 11 created in Micromine using implicit modelling, limited by a bounding box (top) and cookie cut to restrict it to within 30 m of supporting data (bottom)

14.3 Resource Modelling

14.3.1 Data Flagging

Assay data was flagged in Datamine using the diluted vein wireframes, dyke, weathering surfaces, mined solids, and topography with identifying codes assigned to appropriate fields.

The following wireframes were provided by DPMK:

- A model of the late stage, post mineralisation, mafic volcanic intrusives ("dyke") – which was treated as waste.
- Three wireframe surfaces for the domaining of weathering zones:
    - Base of strong oxidation ("SOX")
    - Base of moderate oxidation ("MOX")
    - Base of weak oxidation ("WOX")
- A topography surface, created using an RTK-GPS system that was set up by Spectrum Surveys in 2006.
- Four wireframes representing the existing mined underground workings, as at 31st December 2013 and received by CSA on 15 January, 2014 which included:
    - Mined stopes above the 780 level (*st780+*)
    - Mined stopes below the 780 level - updated Dec 2013 (*st780-*)
    - All development- updated Dec 2013 (*olddev* - Jul2013 and *newdev*).

Table 37. Definition of codes used in pre-estimation data flagging

Field	Values and Definition
GEOLN	100 for Vein, 900 for dyke (unmineralised), 200 for everything outside vein and dyke
VEIN	Vein number –codes identifying all vein clusters.
WEATHN	1000 to 4000 defining fresh to strongly oxidised material
MINZON	1 for mineralised material in veins; 0 for unmineralised dyke or potentially mineralised material outside wireframed veins

14.3.2 Compositing

Sampling intercepts were dominantly 1 m for drill holes, as illustrated in Figure 37, because sampling intervals are 1 m (rather than sampling to geological boundaries). Sampling intercepts for channels were more variable, though still dominated by 1 m (due to the standard practice of sampling to geological boundaries). However, by compositing to 1 m, there was a risk of introducing bias into the sampling dataset due to the variable thickness of veins. A method of dealing with this could be to complete a 2D estimate of veins. However, this was not a feasible option for 158 veins.

Therefore, an effort was made to composite sample data in such a way that balanced the variable thickness of veins against the need to composite to satisfy single sample support requirements of Kriging. The process decided upon was compositing that honoured the minimum mining width of 1.8 m. All samples within the wireframes were composited to down hole lengths that correspond to or close to 1.8 m true thickness. To complete this, the orientation and dips of drill holes and channels had to be accounted for as well as the orientations of the veins. Seven orientation domains were used to inform the dip and dip directions of the veins.

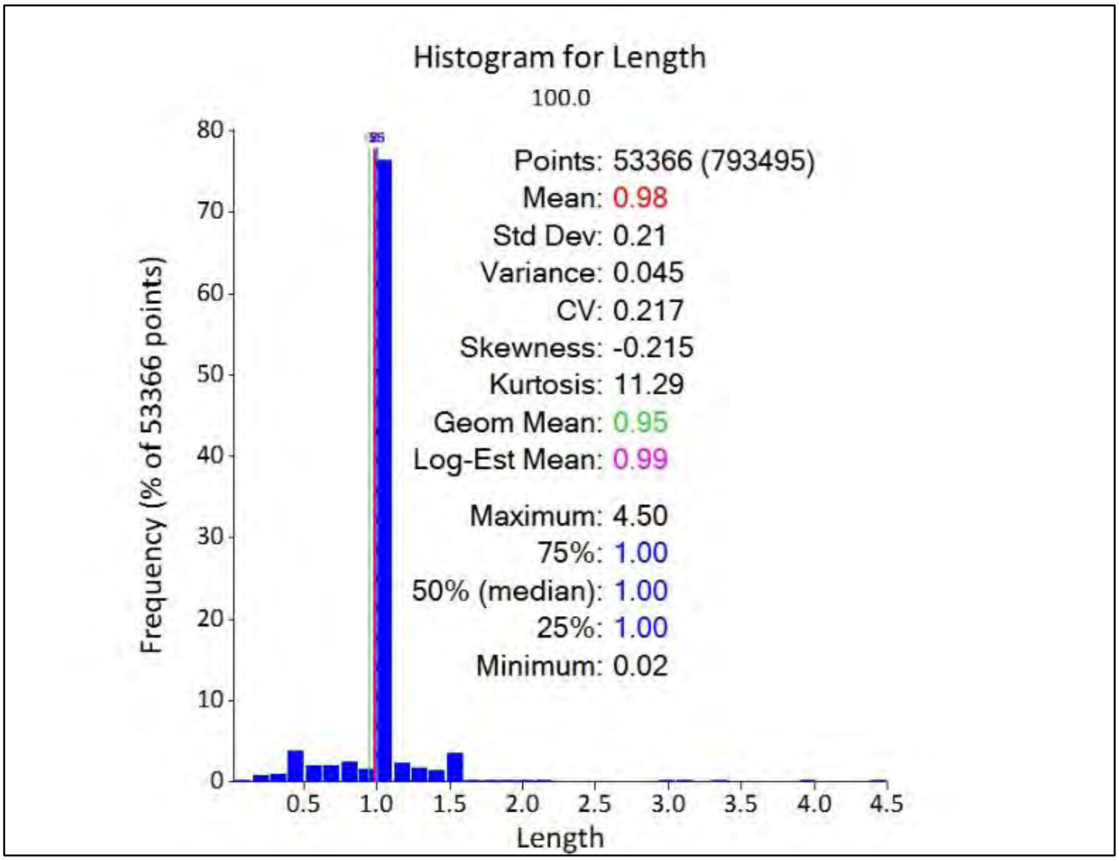


Figure 37. Normal histogram of sample lengths of assays within diluted vein wireframes.

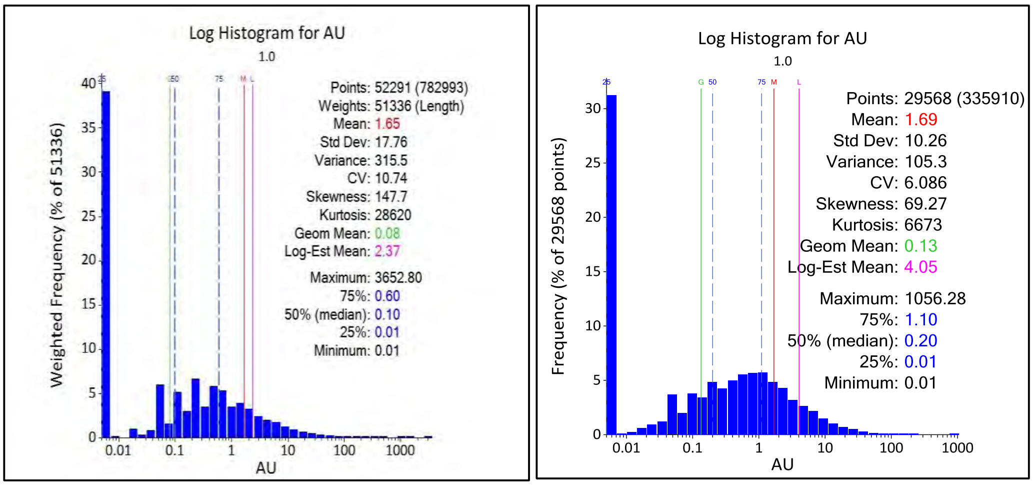


Figure 38. Histogram for gold (Au) for raw non-composited data (left) and data composited to 1.8 m true width (right), both datasets are for MINZON=1.

14.3.3 Domain Descriptive Statistics for the Vein Model

The dataset used for the estimation of the vein model was reviewed in the context of mineralisation and oxidation to define estimation domains. The mineralisation represents both vein and selvedge mineralisation, and due to limitations in the detail of the geological logging at this point, there is limited geology available to further domain. The results were as follows:

- There was no significant oxide or transitional material in the area estimated (<2% of the vein data), therefore these two surfaces were combined during estimation.

- Estimation domains for the data consisted of all sample points that fell within the vein wireframes. This is believed to be justified since veins are often inter-related and comprise of main vein and splay. In addition, there were often insufficient samples within individual veins to estimate so soft boundaries were used between veins. Note: Search ellipses were constrained to limit the undue influence of other veins in the estimation in areas well supported by drilling.

Table 38. Table of summary statistics for the 6 estimated variables in the vein model (MINZON=1)

Statistic	Au	Ag	Cu	Pb	Zn	S
Count	29568	29529	29623	29394	29481	17594
Minimum	0.01	0.50	0.00	0.00	0.00	0.05
Maximum	1056.28	3530.70	26.00	15.50	51.26	41.32
Declustered Mean	1.26	23.48	0.24	0.08	1.01	2.54
Mean	1.69	27.75	0.30	0.12	1.42	1.64
Std Dev	10.26	83.56	0.75	0.50	3.24	3.07
Variance	105.33	6982.39	0.56	0.25	10.47	9.45

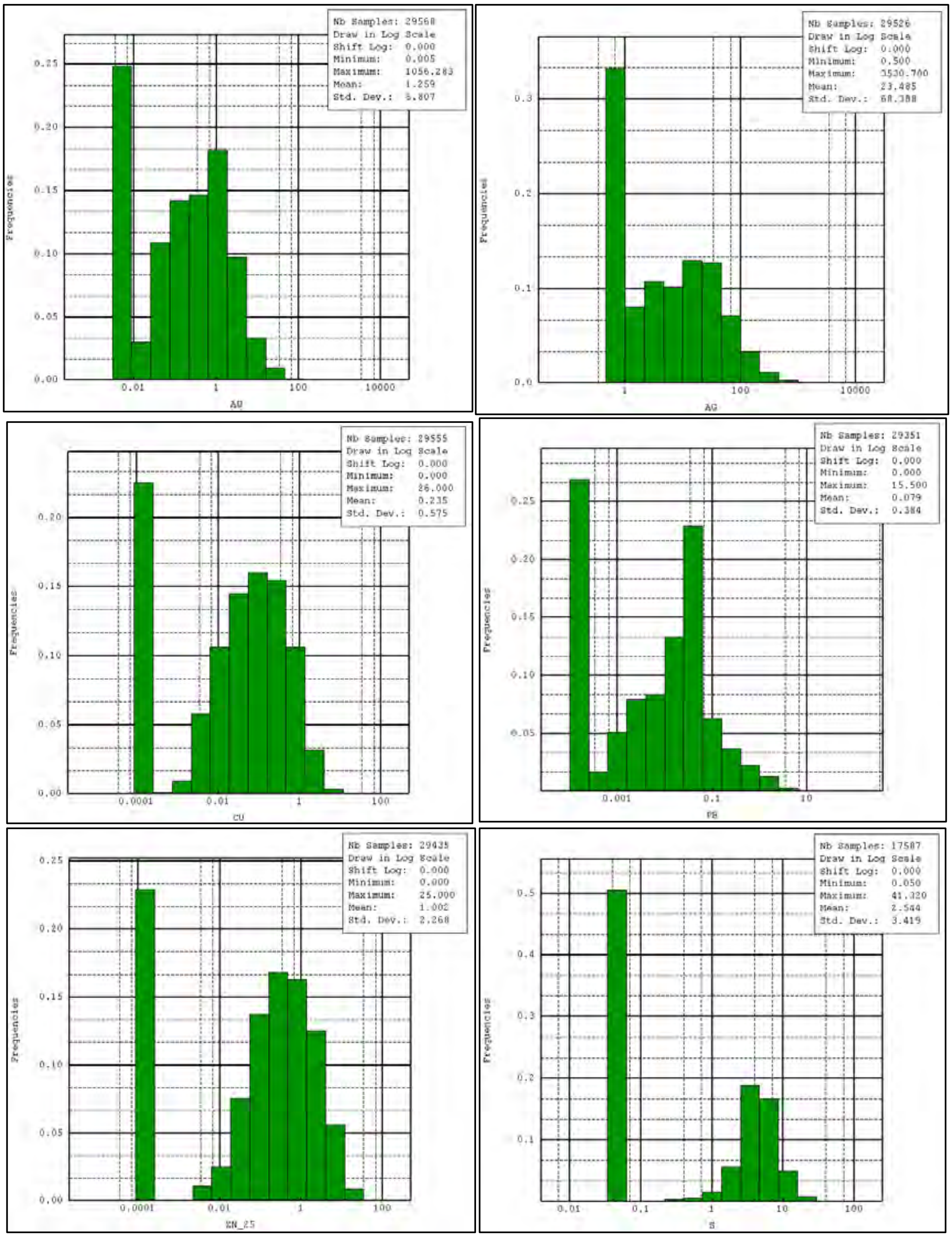


Figure 39. Log histograms for the 6 estimated variables of the vein model (MINZON=1)

14.3.4 Application of Top-cuts for Cu, Zn, Pb and S

Note: In the vein model, outliers for Au and Ag were controlled in a different way to top-cutting, see Section 14.3.5 and Section 14.4.5.

Extreme grade outliers, although real, are not representative of the sample population and therefore need to be controlled to limit their in the estimation process. Top-cuts were chosen for Cu, Zn, Pb and S by reviewing the histograms of the vein model data.

For Cu, Zn, Pb and S, the point at which the histogram broke down was chosen as an appropriate top-cut. During this selection process, the proportion of samples and the influence of the applied top-cut on the global grade were reviewed to assess the material impact of the cut.

Note: to minimise impact and honour local scale variability, high grade outliers (which exceeded a top-cut value) remain uncut for the estimation of blocks with centroids within 1.5 m of the high grade sample.

The top-cuts for Cu, Zn, Pb and S for the vein model are presented in Table 39.

Table 39. Top-cuts used for Cu, Pb, Zn and S used in the Vein Model estimate

Variable	# Samples	Un-cut Mean	COV	Top Cut Value	Cut Mean	Declustered Cut Mean	# Samples Cut	% Samples Cut	% Metal Cut
Cu	29,623	0.302	2.476	6.0	0.296	0.24	75	0.25%	-2.10%
Pb	29,394	0.118	4.204	6.5	0.115	0.08	32	0.11%	-5.80%
Zn	29,481	1.418	2.282	25.0	1.402	1.00	63	0.21%	-1.10%
S	17,594	1.635	1.880	15.0	1.609	2.24	121	0.69%	-1.60%

14.3.5 Variography Study

Kriging was used for the estimation of all grades and therefore, geostatistical analysis was required to derive Kriging models to define the directions and ranges of grade anisotropy in the deposit. Variography was completed in Supervisor software for Cu, Zn, Pb and S. A modified Kriging approach was used to estimate Au and Ag and variograms for these variables were generated in ISATIS software. All variography was completed on uncut, composited data.

Vertical (down the hole) variograms were generated to determine the nugget variance. Nuggets were moderate to high in the majority of cases, for all elements in vein and in probability modelled mineralisation.

A Gaussian transformation was applied to the composites for Cu, Zn, Pb and S which normalised the data, resulting in variograms that are a clearer representation of the spatial characteristics of the underlying distribution. The modelled Gaussian variograms were back-transformed into the original sample space since Gaussian variograms cannot be used as direct inputs into Kriging.

As mentioned in Section 14.3.4, there was a different approach taken in the estimation of grade for Au and Ag in the vein model, which is briefly outlined here to preface the variography methodology that follows.

The method is based on work reported by Rivoirard, detailing what he calls "A Top-Cut Model for Deposits with Heavy-Tailed Grade Distribution" (Rivoirard et al,2012). This proposes a geostatistical model that handles high values without cutting them and is based on the following assumptions for extreme grades:

- There is a point at the extreme grade end of the tail of most grade distributions which represents the grade beyond which, grade behaviour is purely random (the "top-cut" value).

- In such cases, the only tangible information that can be estimated is the geometry i.e. the location of that extreme grade may be estimated, but we are unable to reliably estimate the exact grade of it.

Instead of normal top-cutting, which removes metal from the dataset, no data gets cut from this method. Instead, the grade variable gets split into three parts:

v1 = The data below the "top-cut" whose spatial relationships can be modelled and reliably Kriged using the variogram.

v2 = An indicator variable which defines the data above the "top-cut" (assigned a code of 1) and the data below the "top-cut" (assigned a code of 0).

v3= The mean grade of the "excess" i.e. the mean grade of the data that is above the "top-cut". This is pure nugget since it has no spatial correlation with the grades below the "top-cut". Therefore, in an estimation sense, the only meaningful estimate for this "extreme" grade portion of the grade distribution is the mean grade of the "excess".

The first two variables are co-Kriged since they are correlated, with the final grade of the variable being v1+(v2*v3). Therefore, the metal attached to extreme grades above the "top-cut" is re-distributed in blocks which have been estimated to contain a proportion of this extreme grade material.

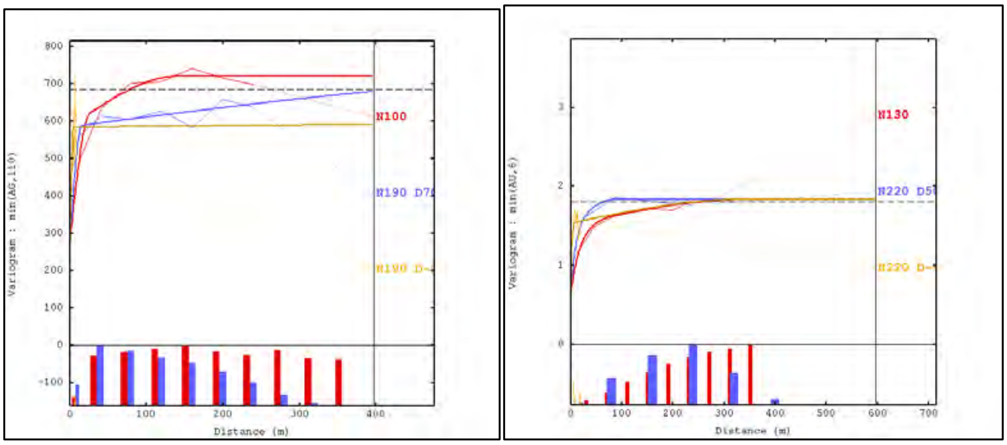
The method is believed to be more robust for heavy tailed grade distributions like that seen for Au at Shahumyan because it allows for the modelling of more robust variograms (that don't mix extreme grades that are behaving randomly), and also does not cut metal from the deposit, which normal top-cutting does. The method was used for Ag to ensure that the strong correlation observed between Au and Ag in the data was maintained in the estimate.

The choice of "top-cut" is made where the structure of the variogram breaks down and becomes pure nugget. This review was completed for Au on the ratio of simple to cross variograms for indicators based on percentiles from 95th to 99.9th. A "top-cut" of 6 g/t was chosen for Au, which corresponded to the 95th percentile. The equivalent percentile value for Ag was chosen (110 g/t).

Table 40. Statistics for Au and Ag for upper percentiles. Chosen “top-cut” values in bold.

Variable	Q95	Q97	Q97.5	Q98	Q99	Q99.9
AU	**6**	9	11	12	19	67
AG	**110**	158	179	206	322	892

Cross-variography was then completed on the Au values below the chosen “top-cut” (6 g/t), and the indicator, so that the two correlated variables could be co-Kriged. Similarly for Ag, (where the “top-cut” value was 110 ppm) and the indicator for Ag, as above. The cross-variogram models for Au are illustrated in Figure 40. Models for all variograms used in Kriging are presented in Table 41.



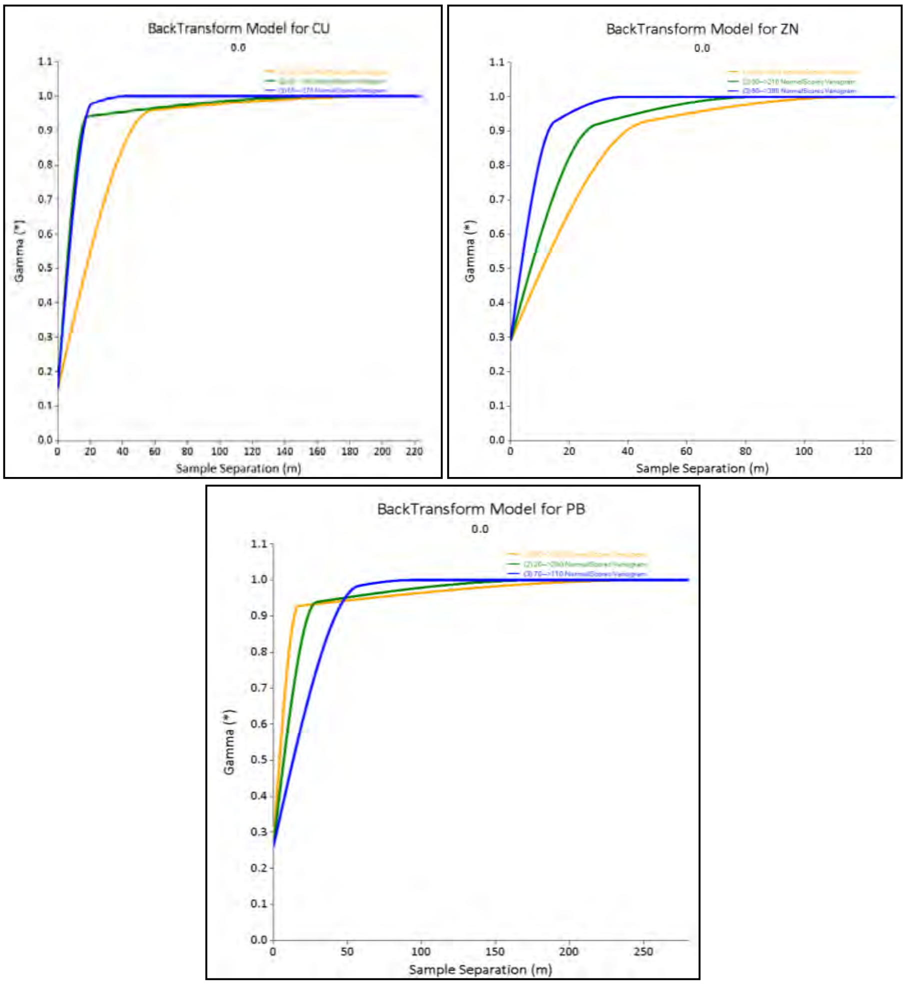


Figure 40: Variogram models for Au, Ag (modelled in ISATIS) and Cu, Zn and Pb (modelled in Supervisor) for Minzon1 data

Table 41. Variogram parameters used in estimation of Au, Ag, Cu, Pb, Zn, S

Variable	Rotation (ISATIS Mathematician Mode)			Nugget	Structure 1				Structure 2			
	Z	Y	X		Sill	Range 1	Range 2	Range 3	Sill	Range 1	Range 2	Range 3
Au<6	-10	0	70	0.35	0.49	55	29	5	0.16	355	86	30
Ind(Au,6)				0.92	0.05				0.03			
Au/Ind				0.66	0.24				0.10			
Ag<110	-40	0	50	0.39	0.42	26	15	5	0.19	143	80	15
Ind(Ag,110)				0.59	0.28				0.13			
Ag/Ind				0.84	0.11				0.04			
CU	-180	30	0	0.15	0.78	58	18	21	0.07	203	169	42
ZN	150	30	0	0.29	0.55	46	29	15	0.32	369	235	102
PB	70	0	20	0.26	0.66	17	29	58	0.08	255	184	95
S	-180	10	0	0.08	0.6	44	55	48	0.32	369	235	102

14.4 Block Modelling

14.4.1 Block Model Extents and Block Size

The volume block model for the vein model was created in Datamine. This was flagged by the vein solids, a wireframe solid of the unmineralised dyke material, the weathering surfaces WOX, MOX and SOX, topography DTM surface and solids representing historic and recent development and mining.

The parent block size was 3 m x 3 m x 3 m (X x Y x Z). This was sub-celled to 0.5 m x 0.5 m x 1 m within mineralised, geology and mined volumes

This sub-celled model was then regularised back to 3 m x 3 m x 3 m, with the addition of three proportion fields (mined material, mineralised vein material, and waste dyke material), and grade estimation was completed in ISATIS.

The block model extents are presented in Table 42.

Table 42. Block model extents for the Kapan MRE

Dimension	Minimum	Maximum	Parent cell size
X	22,800	24,612	3
Y	42,634	45,061	3
Z	252	1,002	3

14.4.2 Dynamic Anisotropy

Dynamic Anisotropy (“DA”) was used to guide the orientation of the search ellipse during estimation in ISATIS to account for variably oriented narrow veins.

Data for the DA was obtained from dip and dip direction of the triangles of the diluted southern surfaces of the diluted wireframes and estimated into the block model in Datamine.

14.4.3 Block Model Attributes

The attributes included in the block model files are presented in Table 43.

Table 43. Block model attributes for the Kapan MRE block model

Attribute	Description
IJK	Independent block identifier
XC	Centroid of the block in the X (easting) direction
YC	Centroid of the block in the Y (northing) direction
ZC	Centroid of the block in the Z (RL) direction

Attribute	Description
XINC	Block extent in the X (easting) direction
YINC	Block extent in the Y (northing) direction
ZINC	Block extent in the Z (RL) direction
AU	Gold value in g/t
AG	Silver value in g/t
CU	Copper value in percent
ZN	Zinc value in percent
PB	Lead value in percent
S	Sulphur value in percent
PASS	Estimation search pass - 1 = first search pass; 2=second search pass; 3=third search pass; Blank search pass meant that average grades per vein were assigned to blocks.
AVGDIS	Average distance of samples (used to estimate) from block centroid
NUMSAM	Number of samples used to estimate
SLOPE	Kriging slope of regression between estimated and true grades
WOM	Weight of the mean
ESTZON	Estimation zone (1-13 = Probability Model, 50 = Vein Model)
GEOLN	Geology Number/Domain: 100 = Vein Wireframe Model, 900 = Dyke Model, 200 = all other material
MINED	Mined proportion field between 0 and 1 where MINED=0 if the block is unmined
MINZON	Proportion of the block that is mineralised. Not modified to account for dyke.
TOPO	Blocks below topography, 1 = below
WEATHN	Weathering field where 1000=Fresh, 2000=Weakly Oxidised, 3000=Moderately Oxidised and 4000=Strongly Oxidised
DYKE	Blocks within the Dyke wireframe model - values of 0 to 1 based on how much dyke in the block
VEIN1	Dominant vein number
VEIN2	Secondary vein number
VNPROP1	Proportion of the block flagged by the dominant vein
VNPROP2	Proportion of the block flagged by the secondary vein
VEIN1A	Alpha field (name) for Vein1
VEIN2A	Alpha field (name) for Vein2
AGAV	Ag mean grade of vein assigned to unestimated blocks
AUAV	Au mean grade of vein assigned to unestimated blocks
CUAV	Cu mean grade of vein assigned to unestimated blocks
PBAV	Pb mean grade of vein assigned to unestimated blocks
SAV	S mean grade of vein assigned to unestimated blocks
ZNAV	Zn mean grade of vein assigned to unestimated blocks
DENSITY	Specific gravity of block assigned to blocks using regression with estimated grades
AU_GKEQ	Gold Equivalent value, based on October 2013 equation
PROFIT_T	Profit Tonnes value
NSR	Net-Smelter Return
CLASS	Class field where 2=Indicated and 3=Inferred

14.4.4 Kriging Neighbourhood Analysis

Kriging Neighbourhood Analysis ("KNA") was completed in ISATIS software in order to optimise the neighbourhoods for Kriging. The criteria reviewed were:

- The slope of the regression of the "true" block grade and the "estimated" block grade
- The weight of the mean – which reflects local variability
- The distribution of the Kriging weights, including the proportion of the negative weights
- The Kriging variance.

The results informed the choice of search ellipse ranges, and sample search criteria in the first search pass for grade estimation, as presented in Table 45.

The block size was chosen to align with mine planning requirements on-site, and taking into account the close spaced data in production areas. However, the size is small relative to drill spacing in outlying parts of the deposit, which has resulted in sub-optimal Kriging statistics in such areas.

14.4.5 Grade Estimation

Ordinary Kriging was used to estimate grade for Cu, Zn, Pb and S into the 3 m x 3 m x 3 m vein block model.

Soft boundaries were used between veins. Many individual veins were small, and would not have contained enough data to estimate using hard boundaries. However, for the larger veins, neighbourhood search parameters were such that any data from other veins used to inform blocks would be likely limited to later search passes.

Search ellipse ranges are presented in Table 45. Seven search passes were used to estimate grade into the vein model; three passes using short ranges were used to estimate blocks where there are channels and clustering. These tighter search passes were employed to improve the estimate of grades in those areas and to reduce the influence of close-spaced data in areas of wider spaced drilling. A further four were used for blocks informed by wider spaced drilling.

If blocks did not get estimated in the seven search passes, the mean grades of the data in that particular vein were assigned to the block. This was completed because while there is sufficient confidence in the wireframe interpretation to confirm mineralisation, there is insufficient data to estimate a reliable grade. In these cases, the domain grade is considered the most reliable grade. A total of 2% of the estimated blocks at no cut-off received the mean grade, an approximately 3% of the blocks reported above the resource cut-off received the mean grade.

The dip and dip directions of blocks, based on Dynamic Anisotropy described in Section 14.4.2, were used to create localised search rotations for the ellipse in the model.

As previously discussed in Section 14.3.5, the grades for Au and Ag in the vein model were estimated using the Top-Cut Model for Deposits with Heavy-Tailed Grade Distribution.

6 g/t for Au and 110 g/t for Ag were the grades identified above which there was no spatial correlation between grades.

The truncated variable for Au (Au<6) was co-Kriged with the Indicator for Au at that top-cut. The mean grade of the excess (i.e. that part of the grade distribution above 6 ppm) was noted. This resulted in an estimate of the truncated variable, the proportion of a block above 6 ppm which were used as inputs to the following equation:

$$Au\_OK = E(Au<6) + (E(Ind \geq 6) * \text{Mean grade of data} > 6)$$

Where:

Au_OK = Final Au grade of block

E(Au<6) = Estimated Kriged grade of the truncated variable (grades < 6 ppm)

E(Ind≥6) = Estimated proportion of a block where Au> 6 ppm

Mean grade of data > 6 = 11.81 (describes the part of the distribution where spatial grade behaviour is pure nugget).

Similarly, the truncated variable for Ag (Ag<110) was co-Kriged with the Indicator for Ag at that top-cut. The mean grade of the excess (i.e. that part of the grade distribution above 110 g/t) was noted. This resulted in an estimate of the truncated variable, the proportion of a block above 110 g/t which were used as inputs to the following equation:

$$Ag\_OK = E(Ag<110) + (E(Ind \geq 110) * \text{Mean grade of data} > 110)$$

Where:

Ag_OK = Final Ag grade of block

E(Ag<110) = Estimated Kriged grade of the truncated variable (grades < 110 ppm)

E(Ind≥110) = Estimated proportion of a block over 110 ppm where Ag> 110 ppm

Mean grade of data > 110 = 144.55 (describes the part of the distribution where spatial grade behaviour is pure nugget).

Au Equivalent was back-calculated into the model on completion of the estimate using the formula provided by DPMK:

$$AuEq=Au+(Cu\times1.20)+(Ag\times0.02)+(Zn\times0.34)$$

Metal prices used in the AuEq calculation are presented in Table 44. No recoveries were used.

Table 44. Metal prices and recoveries used for AuEq

Metal	Price USD	Unit
Au	1,250	oz
Cu	2.75	lb
Ag	23	oz
Zn	0.85	lb

Table 45. Sample search parameters for all search passes.

Search Pass	Search Radii			Samples used per estimate		
	X	Y	Z	Minimum	Angular Sectors	Maximum Samples
1	10	10	4	16	8	40
2	15	15	4	16	8	40
3	30	30	4	4	1	12
4	70	50	5	18	8	40
5	110	85	8	12	8	40
6	140	100	8	4	1	8
7	300	200	10	4	1	8

14.5 Bulk Density

As described in Section 12.3, due to limitations in the detail of geological logging, it was not possible to assign higher densities to that part of the mineralisation represented by veins. Instead, a regression of density against AuEq grade was used to model variability of density within the block model.

To obtain the regression, all density data was flagged by the vein model (MINZON1) and the subset was used to determine the regression. Outliers at the 1st and 99th percentile were excluded from the fit. Data was plotted in a natural log space and linear and polynomials trends were reviewed by:

a) Comparing the r^2 (goodness of fit) in excel

b) Visualising the fit of the trendline at different grade cut-offs

c) Subsetting MINZON1 data and comparing results across subsets.

For all 4th order polynomials and higher, r^2 exceeded 0.995. Based on this the simpler 4th order polynomial was selected:

$(y) = EXP\ [((x^4)\text{* } 00026921)+((x^3)\text{* } 00253321)+((x^2)\text{* } 00305893)+(x\text{* } 01235032)+ 00333878]$

Where Y = calculated density, and X=grade (AuEq).

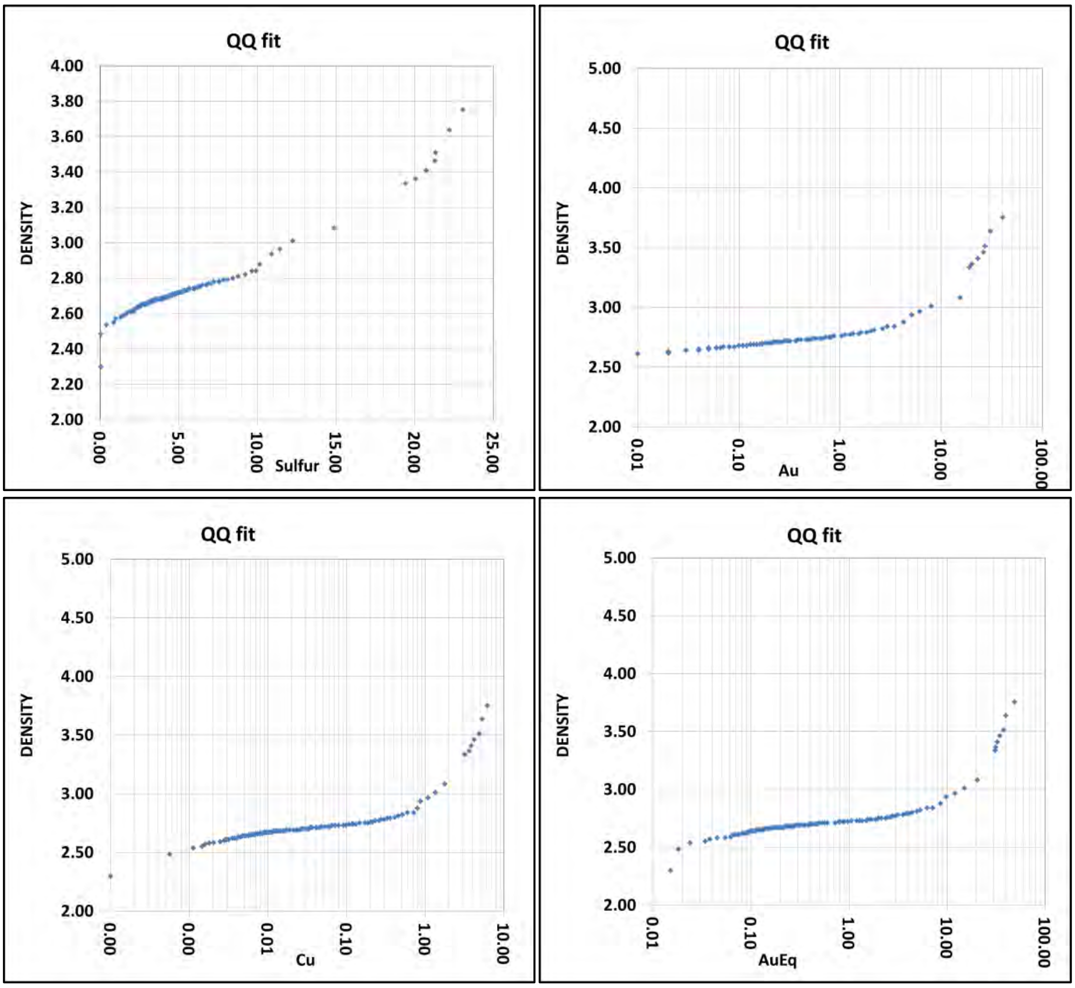


Figure 41. QQ plots showing relationship between Sulphur, Au, Cu, Au_GKEQ and density (clockwise from top left).

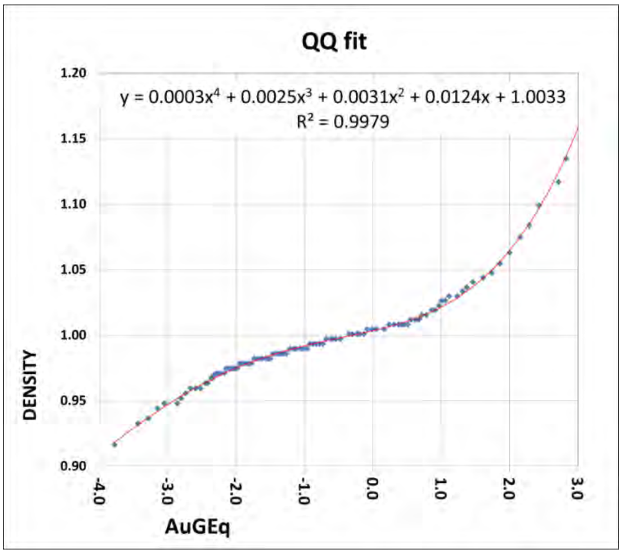


Figure 42. 4th order polynomial regression selected to define density relationship following the estimation of AuEq.

14.6 Block Model Validation

The vein model was exported from ISATIS following grade estimation and imported into Datamine where it was validated, classified and reported.

Validation on the output MRE model was undertaken for Kapan, and consisted of a review of the following:

- Visual check of cross sections; comparing the output model (coloured by grade) versus input samples coloured by grade, as illustrated in Figure 44 to Figure 46.
- Comparison on a global scale; of histograms and probability plots of the input sample grades against the output block model grades, as illustrated in Figure 47 and Figure 48.
- Swath plots for three directions (easting, northing and elevation), which compares at a semi-local scale; model tonnes versus drill hole metres, and composite grades versus model grades, as illustrated in Figure 49.
- Checks on the individual mean grades of some larger veins versus the mean grades of the blocks for those veins.
- Reporting of model at a variety of cut-offs to evaluate the impact of the density regression, presented in Table 46.

Validation confirmed that the block model reflected the tenor of input grade and can be considered reliable on a local scale in the Indicated areas, and reliable on a global scale in the Inferred areas.

Cross sections of the resource were reviewed to assess if the grades of reflect the grades of the composites. These validated well, and three example sections are illustrated in Figure 44 to Figure 46.

The histograms for the Au and Cu estimates and input data are illustrated Figure 47 and Figure 48. The high variability at short distances results in a high degree of smoothing in the model which is reflected in the histograms. The mean grades of inputs versus outputs compare reasonably well for all major elements.

For the elements that contribute to Au Eq, estimated grades for Au, Ag, and Cu compared closely with the input data. However, Zn was 19% lower in the block model than in the input data. It is currently unclear why this is the case, however, this will be reviewed in advance of the next MRE update.

Table 46. Comparison of block model grades versus inputs

Variable	Block Model Grade	Composite Grade (Declustered)	% Difference
Au	1.30	1.26	3%
Ag	23.70	23.49	1%
Cu	0.24	0.24	2%
Zn	0.81	1.00	-19%
Pb	0.07	0.08	-13%
S	2.21	2.54	-13%

Swath plots illustrated in Figure 49 were generated to validate the block model on a semi-local basis, showing the trends in input data (grades, number of drill hole metres) compared against trends in the model data (grades, tonnages) along slices of eastings, northings and benches.

These show that the model grades broadly honour the trends and the tenor of the inputs. The model grades are smoothed compared to the input grades. This is expected due to the methodology used but is not considered to be excessive in areas of current and proposed production. The best correlation between grades occurs where there is the most amount of drilling, with the edges of the deposit seeing the least amount of correlation due to insufficient sample support.

Validation of the density regression in the block model following estimation was undertaken by:

a) Comparing calculated densities versus flagged densities within 20 of the largest veins, which had the highest proportion of flagged density data and approx. 50% of the resource by volume.

b) Comparing a block model report at a suite of AU Eq cut-offs using the regression density against a constant MINZON1 density of 2.73 g/cm^3 as used in the January 2013 MRE (CSA,2013[2]).

In summary:

- The average regression density for the larger veins was within 1% of the RAW flagged density data within these veins.
- Reporting of the block model showed -1% tonnes at a zero AuEq cut off however greater tonnes (between 1 and 2%) for >3 g/t Au Eq cut-off, as presented in Table 47.

Although the use of a regression for the calculation of density is appropriate and considered an improvement over a single average applied to the entire vein mode, validation work shows that further work is necessary to improve the dataset from which the regression is formulated. This is clear since the grade distribution of the dataset used to formulate the regression polynomial, does not reflect that estimated within the vein model, as illustrated in Figure 43.

Table 47. Block Model Report comparing tonnes and metal differences between the regression density versus the constant density (2.73 g/cm³) used in January 2013 MRE (CSA,2013[2]).

BM Report Cut-off (AuEq)	Tonnes Difference	Metal Difference
0.00	-1%	-1%
2.00	0%	0%
2.25	0%	0%
2.50	0%	0%
3.00	1%	1%
5.00	2%	2%
TOTAL	-1%	0%

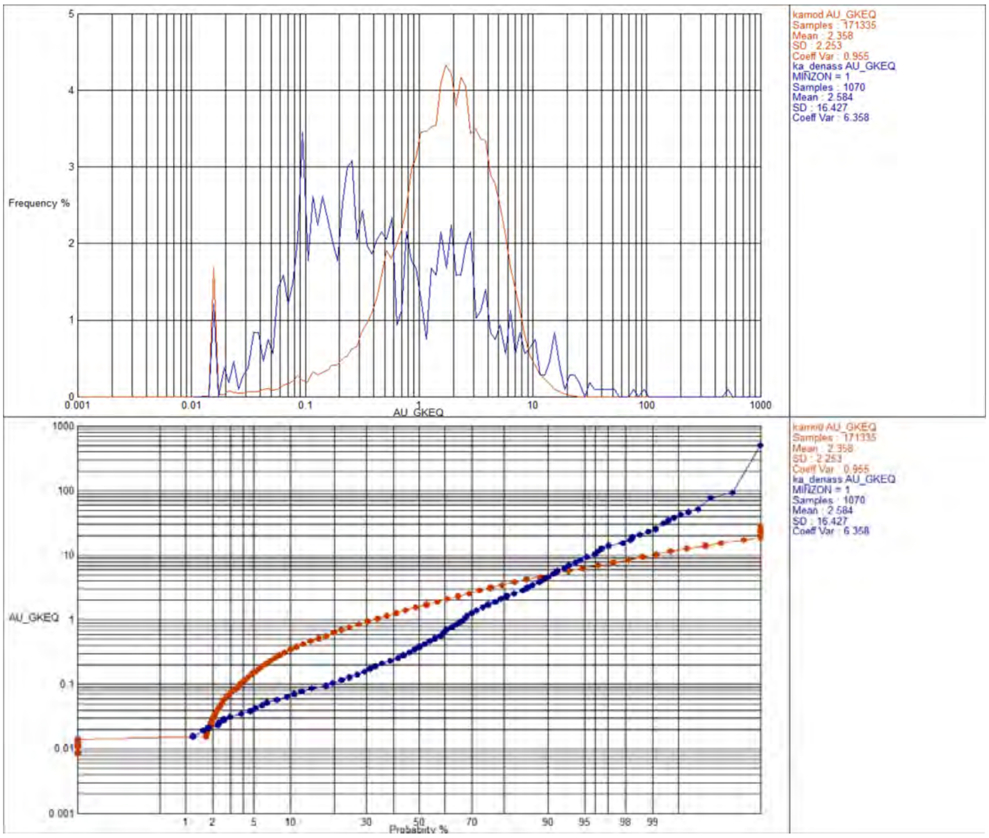


Figure 43. Distribution of modelled Au Eq (red) versus desurveyed density/assay data (blue) used to formulate the density regression polynomial.

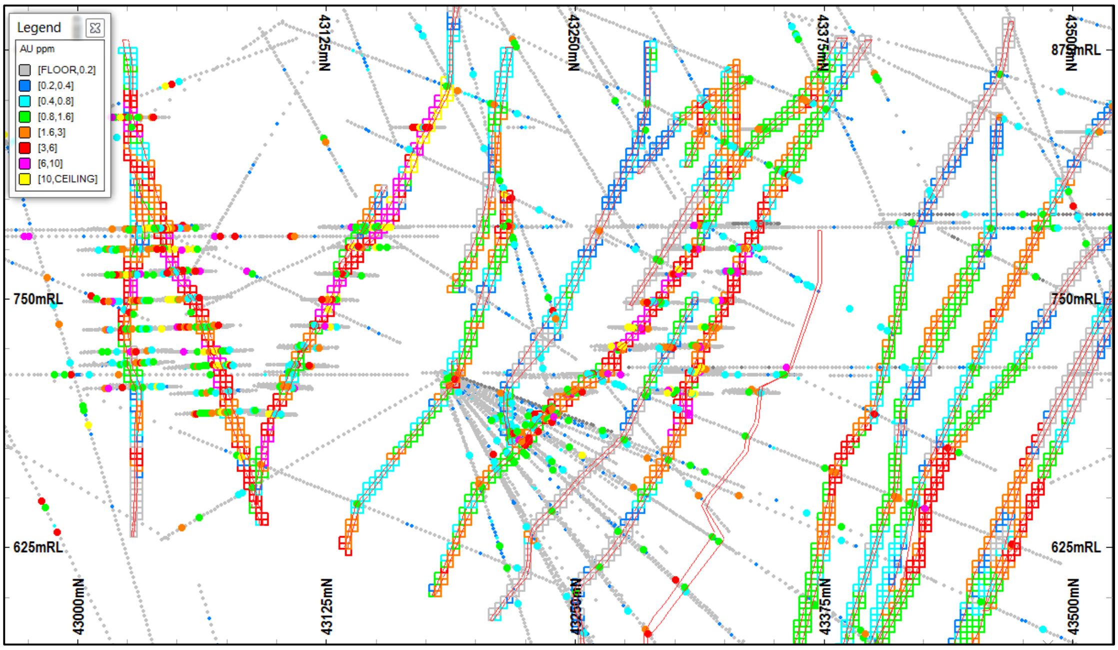


Figure 44. 3x3x3 m regularised vein model, north-south section looking west at 23,800 mE. This shows input drilling composite samples against the output block model, coloured by Au grade. Wireframe solid shown in red.

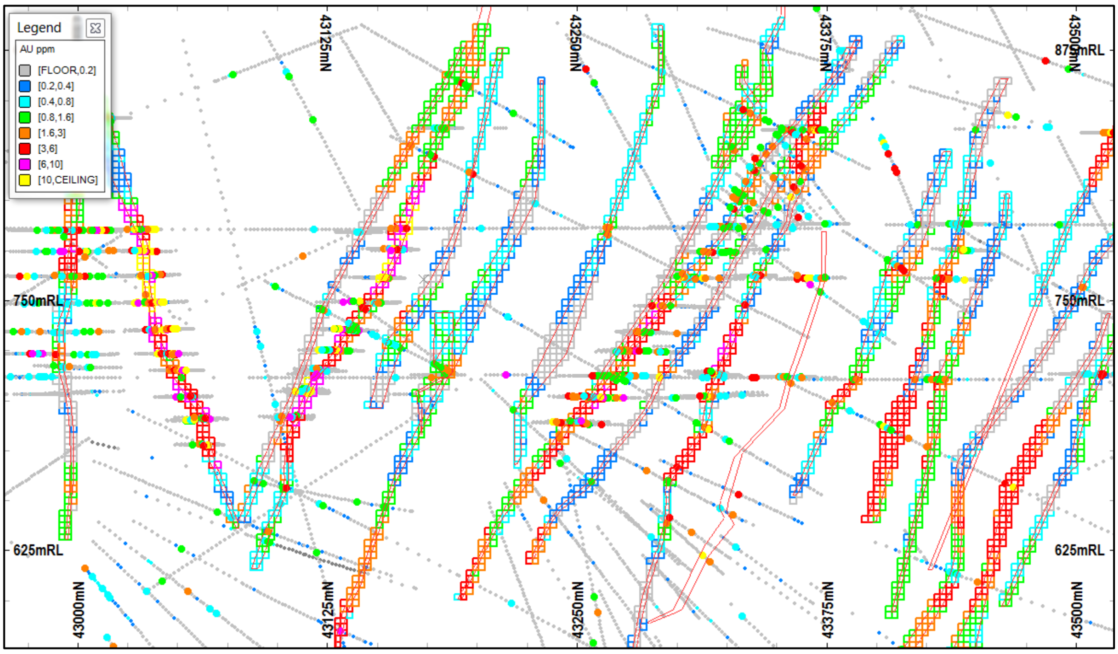


Figure 45. 3x3x3 m regularised vein model, north-south section looking west at 23,850 mE. This shows input drilling composite samples against the output block model, coloured by Au grade. Wireframe solid shown in red.

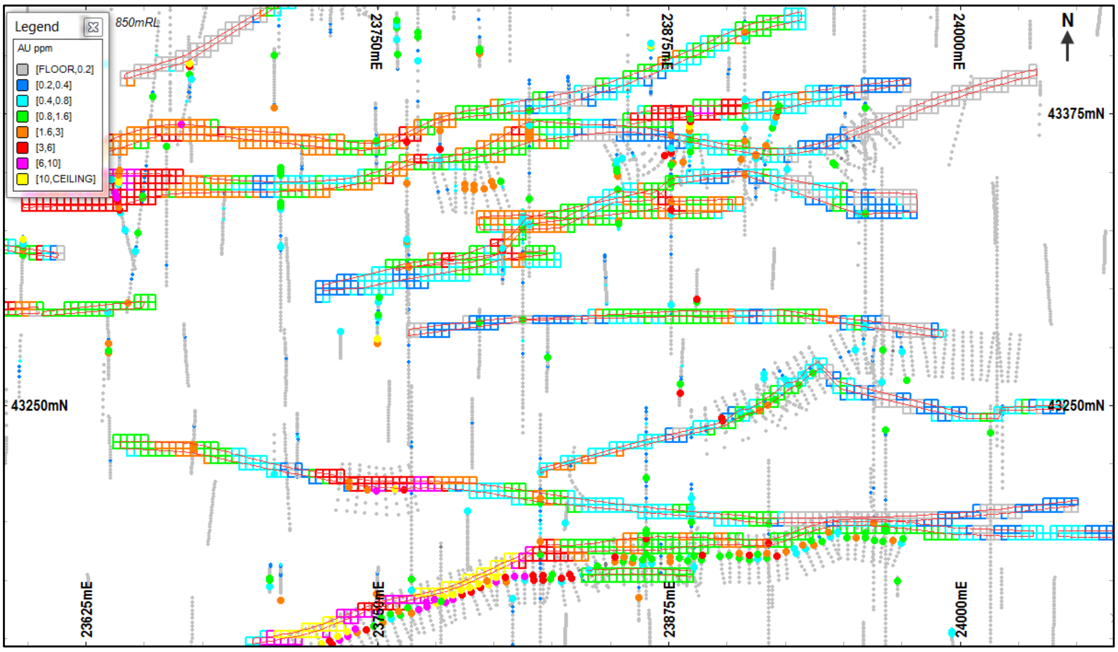


Figure 46. 3x3x3 m regularised vein model, plan view at 850 mRL.
This shows input drilling composite samples against the output block model, coloured by Au grade. Wireframe solid shown in red.

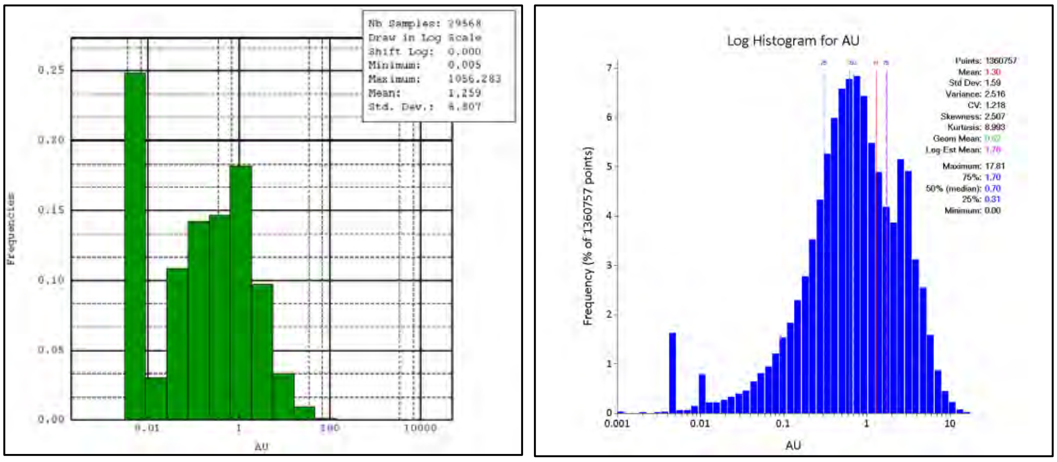


Figure 47. Gold (Au) Log histograms for: declustered MINZON1 composites (left) vs Model (right)

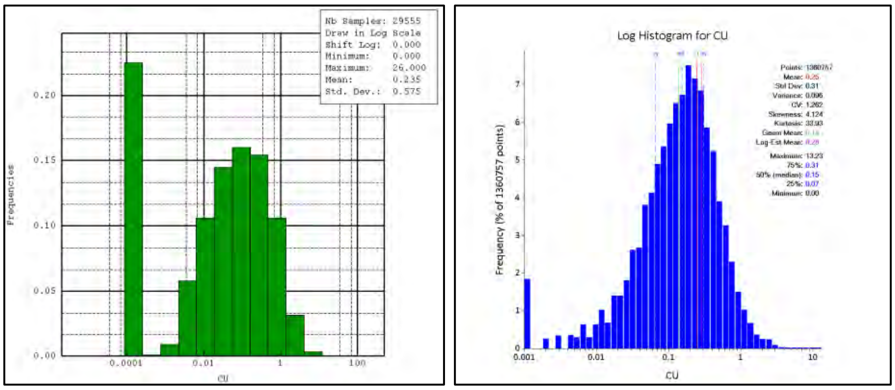


Figure 48. Copper (Cu) Log histograms for: declustered MINZON1 composites (left) vs Model (right)

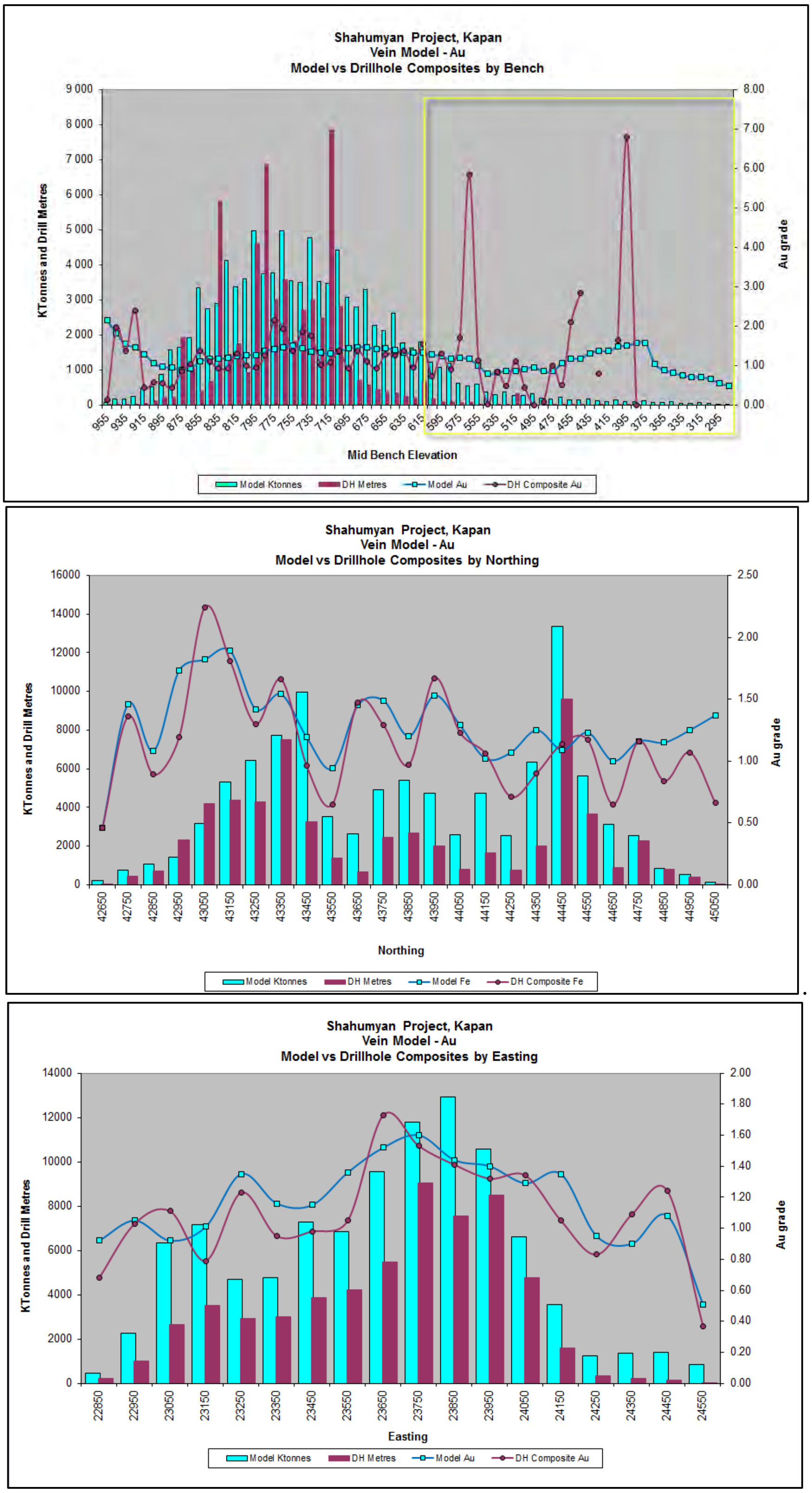


Figure 49. Vein Model de-clustered swath plots, for gold (Au), by Northing, Easting and Bench (top to Bottom).

14.7 Mineral Resource Reporting

14.7.1 *Resource classification*

The Mineral Resource has been classified as Indicated and Inferred based on CIM Guidelines adopted under the Canadian National Instrument 43-101 ("NI 43-101"). Classification was based on a number of criteria, including assessment of the reliability of geological, sample and drill data, and location of production.

Figure 53 illustrates the part of the mineral resource that is classified as Indicated and Inferred. Figure 54 illustrates the block model coloured by Au (g/t), at the reported Au Equivalent cut-off grade of 2.24 g/t.

CSA has considered the following in classifying the MRE:

- Adequate validation of tenement title, drilling, sampling, and geological process completed during a site visit by Mr Galen White, Principal Geologist, CSA Global (UK) Ltd, in February and May 2013.
- Adequate geological evidence for continuity of mineralisation at the cut-off grade used in the estimation of the mineral resource.
- Adequate analytical evidence of gold, silver, copper, and zinc mineralisation.
- Adequate QA/QC controls in place to validate the gold, silver, copper, and zinc grades.
- Adequate diamond core sampling to determine the dry in situ bulk density in order to estimate the tonnage of mineralisation.
- Reliability of interpretation of geological controls, variography and estimation search parameters.
- Review of Kriging statistics – slope of regression, weight of the mean, average distance of samples used to estimate a block and the number of samples used to estimate a block.
- Validation of the resource estimate including review of swath plots to assess the reliability of the grade estimate on a semi-local scale.

Measured Mineral Resources have not yet been classified at Shahumyan for the following reasons:

- It is the opinion of the QP, that reconciliation of the mineral resource against production data and grade control model is required before there is a possibility of classifying any part of the mineral resource as Measured.

Mineral Resources were classified as Indicated largely on the basis of existing production. Mineral Resources in the south of the deposit (at northings less than 44,000 mN) were required to be estimated in the first five passes (the first three passes were used to control clustering in production areas and limit the influence of production data, and the first two in resource development areas) and meet the following criteria:

- If veins have been developed / stoped above the 690 mRL, veins above 660 mRL have been classified as Indicated (Indicated projected 2 sub-levels below development / stopes). Wireframes were created to identify blocks in this area, as illustrated in Figure 50.

- Larger veins with sample support were classified as Indicated above the 640 mRL. These veins were 6, 10, and 11

- Other veins with sufficient sample support were classified as Indicated above the 650 mRL. These veins were 14, 15, 17 and 20.

- The average distance of Indicated blocks is approximately 30 m, which approximates the range of influence of the variograms of major elements.

- The Au variogram model shows >80% of the variability is within the first 30 m (Figure 51) This is reflected in the difference in average sample distances between Indicated and Inferred material within the block model, as illustrated in Figure 52.

- Indicated Mineral Resources in the northern part of the deposit (at northings greater than 44,000 mN) were required to be estimated in the first five passes (first three in production areas, and the first two in resource development areas) and meet the following criteria:

- If veins have been developed / stoped above the 720 mRL, veins above 690 mRL were classified as Indicated (Indicated projected 2 sub-levels below development/stopes). Wireframes were created to identify blocks in this area.

Mineral Resources were classified as Inferred for material that did not meet the Indicated classification criteria listed above. In addition, if blocks were flagged by the Indicated wireframes and were estimated within the first five passes, but belonged to smaller veins, where evidence of continuity of mineralisation and grade is lower they were also classified as Inferred Mineral Resources. These veins are presented in Table 48.

Search pass criteria which strongly influenced the classification of material types were based on the variograms for the major elements.

Table 48. Veins Classified as Inferred Mineral Resources

Inferred Veins	
14	13A
28	13_2
35	17W
4_1	20_4
5X	26_1
6A	27A
10A	31A
11A	35B

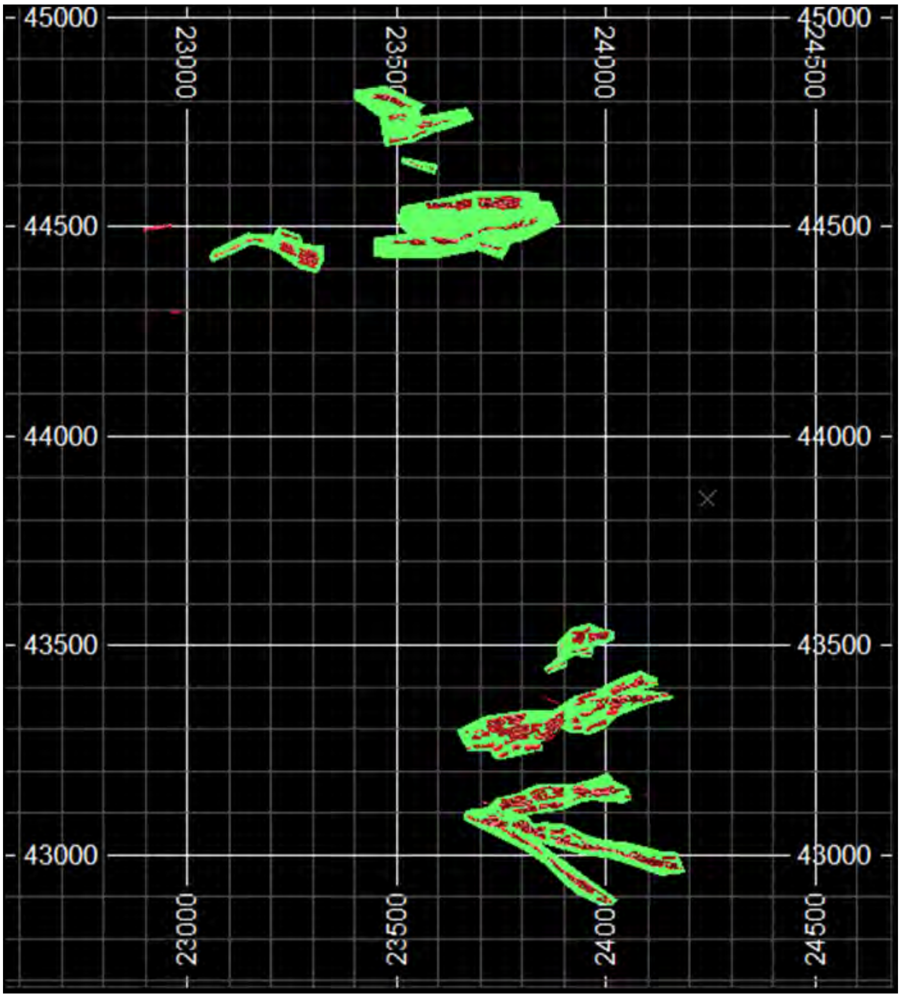


Figure 50. Plan view of wireframe (green) used to isolate mined areas used in the classification of Indicated material

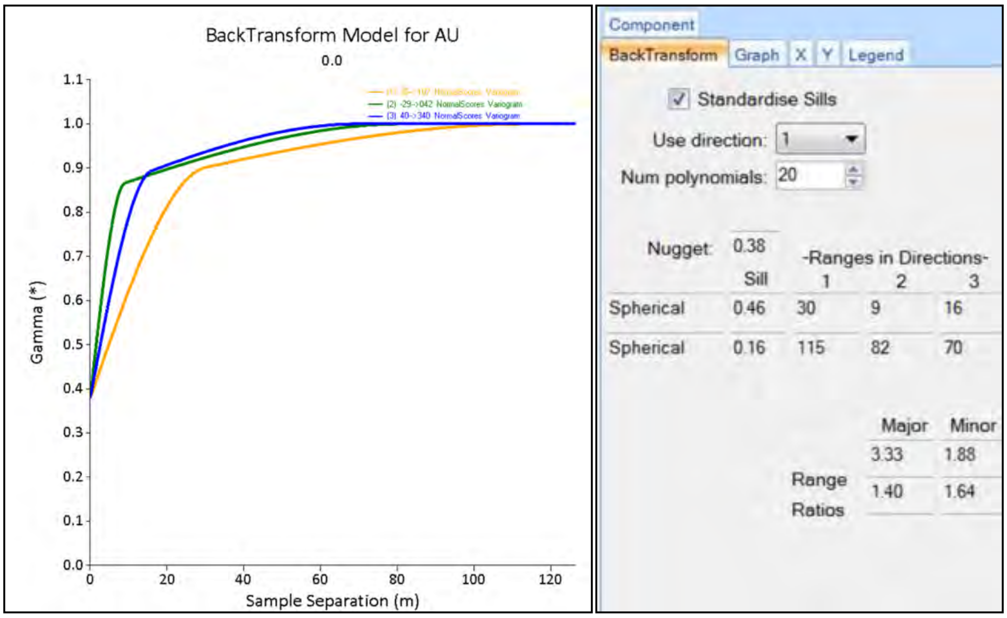


Figure 51. Au variogram model, showing that >80% of gold variability is tied up in the first 30 m supporting the search parameters used in the classification of Indicated.

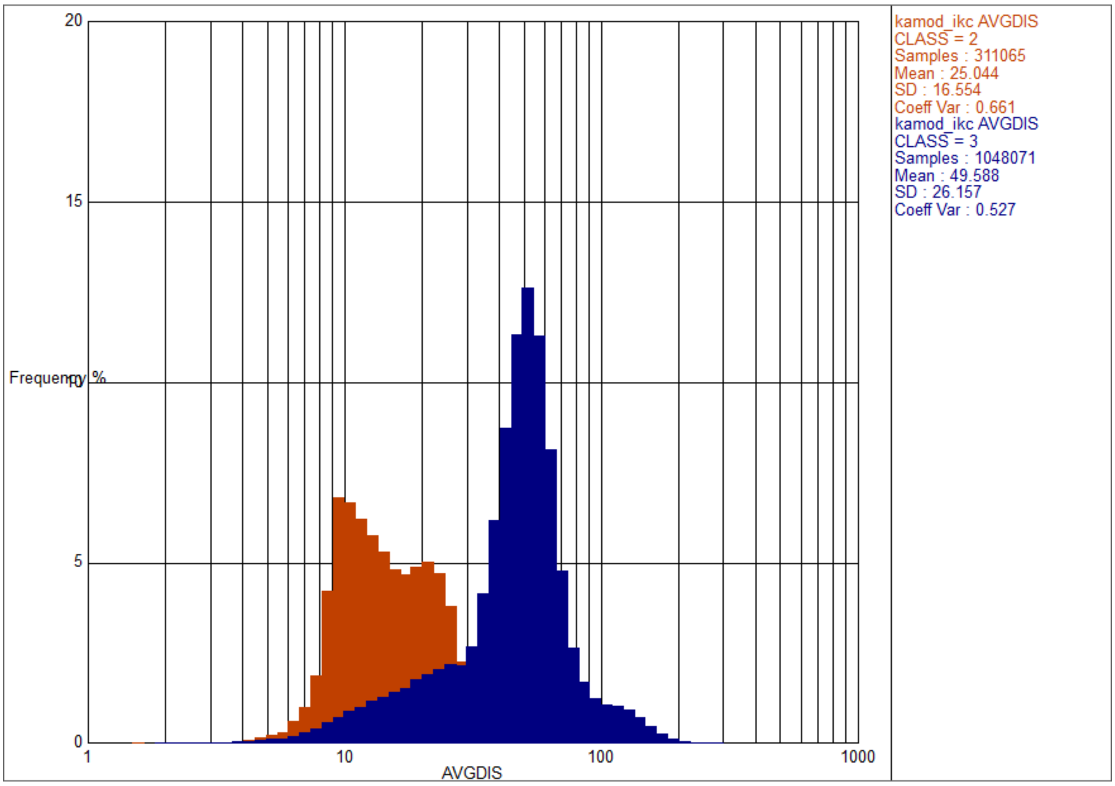


Figure 52. Average distance of samples used to inform a block based, Indicated (blue) and Inferred (red). Note change of support at 30 m spacing.

14.7.2 Resource Tabulation

The MRE for the Shahumyan Deposit is reported at a gold equivalent cut-off of 2.24 g/t and a "conceptual" profitability cut-off (discussed below and presented in Table 49, of > USD 0 and is presented in Table 50. The cut-off grade was selected in light of the economic sensitivities of the project and informed by current mining.

The Kapan MRE is reported at a profit cut-off of >USD 0 and a gold equivalent cut-off of 2.24 g/t.

The gold equivalent values were calculated by using the following formulae:

$$AuEq=Au+(Cu\times1.2)+(Ag\times0.02)+(Zn\times0.34)$$

The profitability value is calculated by the following set of formulae, and represents a conceptual profitability of each individual block, when considerations such as metal price and current recovery are taken into account, derived from current operational experience. It should be noted that the input parameters used in this formula have not been generated following any formal NI 43-101 compliant Economic Assessment, which has not been undertaken, and mineral reserves have not been defined.

There has been inadequate analysis at the project to report mineral reserves; however the mine is currently operating and producing saleable product. The reporting of the mineral resource estimate using both a grade cut-off and the conceptual profitability cut-off has been completed only to justify "reasonable chances of eventual economic extraction" under CIM guidelines, and does not imply the completion of any Economic Assessment, which has not been undertaken.

The two formulas used in its derivation are:

$$Density = EXP\ [((x^4)*\ 00026921) + ((x^3)*\ 00253321) + ((x^2)*\ 00305893) + (x*01235032) + 00333878]$$

Where x = AuEq

$$Tonnes = XINC\times YINC\times ZINC\times(1\text{-}MINED)\times Density\times VEIN$$

Where XINC is the block size in the X direction, YINC is the block size in the Y direction, ZINC is the block size in the Z direction, MINED is the remnant proportion field taken from the mined out areas, and VEIN is the vein proportion field.

$$CuRe=79.84$$

$$AuRe=76.30$$

$$AgRe=68.29$$

$$ZnRe=65.59$$

Where CuRe is copper recovery, AuRe is gold recovery, AgRe is silver recovery, and ZnRe is zinc recovery.

$$CuMetR=(CuRe/100)\times Tonnes\times(Cu/100)\times 2204.6226$$

$$AuMetR=(AuRe/100)\times Tonnes\times(Au/100)\times 31.103477$$

$$AgMetR=(AgRe/100)\times Tonnes\times(Ag/100)\times 31.103477$$

$$ZnMetR=(ZnRe/100)\times Tonnes\times(Zn/100)\times 2204.6226$$

Where Cu is the estimated copper value, Au is the estimated gold value, Ag is the estimated silver value, and Zn is the estimated zinc value. Then:

$$Revenue=(CuMetR\times 2.75)+(AuMetR\times 1250)+(AgMetR\times 23)+(ZnMetR\times 0.85)$$

If Revenue is greater than zero, then the following sub formulae are applied:

$$CuCdmt=(CuMetR/2204.6226)/0.2$$

$$CuCwmt=CuCdmt\times 1.08$$

$$ZnCdmt=(ZnMetR/2204.6226)/0.59$$

$$ZnCwmt=ZnCdmt\times 1.08$$

This all then feeds into:

$$CuTCRC = (CuMetR*0.11) + (0.75*(AuMetR*5.0)) + (0.73*(AgMetR*0.5)) + (CuCdmt*262) + (uCwmt*146)$$

$$ZnTCRC = (ZnCdmt*195) + (ZnCwmt*116)$$

$$OPEX = Tonnes\times 68$$

$$Royalty = (4+(Revenue-OPEX)\times 100)/(Revenue\times 8)$$

$$RoyRev = Royalty\times Revenue\times 0.01$$

Then, if RoyRev is greater than zero, the following applies:

$$\text{"PROFIT"} = (Revenue - (cuTCRC + znTCRC) - OPEX -RoyRev)/Tonnes$$

If RoyRev is less than or equal to zero, then the following applies:

$$\text{"PROFIT\_T"} = PROFIT/Tonnes$$

Inputs in to this formula are presented in Table 48.

Table 49. Inputs to the profitability Calculation – Current Mining Inputs (2013)

<table>
<tr><th>Parameter</th><th>Value</th></tr>
<tr><td>Copper Recovery</td><td>79.84%</td></tr>
<tr><td>Gold Recovery – copper concentrate</td><td rowspan="2">76.30%</td></tr>
<tr><td>Gold Recovery – zinc concentrate</td></tr>
<tr><td>Silver Recovery – copper concentrate</td><td rowspan="2">68.29%</td></tr>
<tr><td>Silver Recovery – zinc concentrate</td></tr>
<tr><td>Zinc Recovery</td><td>65.59%</td></tr>
<tr><td>Refining charges for Copper concentrate</td><td>USD 262/DMT</td></tr>
<tr><td>Refining Charges for other metals in Copper concentrate</td><td></td></tr>
<tr><td>Refining Charges Zinc Concentrate</td><td>USD 195/DMT</td></tr>
<tr><td>Refining Charges for other metals in Zinc concentrate</td><td>USD 0</td></tr>
<tr><td>Cash operating cost long term average</td><td>USD 68/tonne</td></tr>
<tr><td>Royalty rate</td><td>USD 4 + (REVENUE-OPEX)*100)/(REVENUE*8)</td></tr>
<tr><td>Selling Price of Copper</td><td>USD 2.75/lb</td></tr>
<tr><td>Selling Price of Gold</td><td>USD 1250/oz</td></tr>
<tr><td>Selling Price of Zinc</td><td>USD 0.85/lb</td></tr>
<tr><td>Selling Price of Silver</td><td>USD 23/oz</td></tr>
</table>

The selling price inputs in to the profitability calculation were those defined in 2013. It should be noted that the conceptual economic model prepared as part of the PEA (see Section 22) uses modified selling prices based on information and benchmarking reviewed in 2014. CSA does not consider this disconnect to be material or significant to the reporting of the MRE. The profitability cut-off defined during the MRE preparation to define Mineral Resources of reasonable chances of eventual economic extraction is lower than that defined using updated selling prices during the subsequent conceptual economic analysis performed under the PEA.

The MRE is reported as at 31 December 2013, with mined wireframes provided to CSA by DPMK to deplete the resource from the same date.

Mineral Resources estimated for Shahumyan and detailed in this Technical Report were prepared in accordance with the requirements as defined by the CIM Definition Standards June 30, 2011 and National Instrument 43-101 – Standards of Disclosure for Mineral Projects, Form 43-101F1 and Companion Policy 43-101CP.

Table 50. Classified Mineral Resource Estimate for Shahumyan Deposit

Dundee Precious Metals - Kapan. Shahumyan Mineral Resource Estimate as at 31 December, 2013												
Resource Category	MTonnes	**Grades**							**Metal Content**			
		AuEq* (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	S (%)	Au Moz	Ag MOz	Cu MLbs	Zn Mtonnes
Indicated	3.0	5.33	3.21	49.88	0.44	1.77	0.19	2.08	0.3120	4.8480	29	0.0535
Inferred	9.5	4.62	2.83	43.34	0.43	1.21	0.10	3.13	0.8626	13.2100	90	0.1147

MRE is reported using a gold equivalent cut-off of 2.24g/t and an operating net profit cut-off of > USD 0

tonnages are rounded to the nearest 1,000 tonnes to reflect this as an estimate

metal content is rounded to the nearest 100 tonnes or 100ozs to reflect this as an estimate

**AuEq = AuEQ [Au + Cu*1.20 + Ag*0.02 + Zn*0.34]*

To the best knowledge of CSA, the stated mineral resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues that prevent this mineral resource from reasonable prospects for economic extraction. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

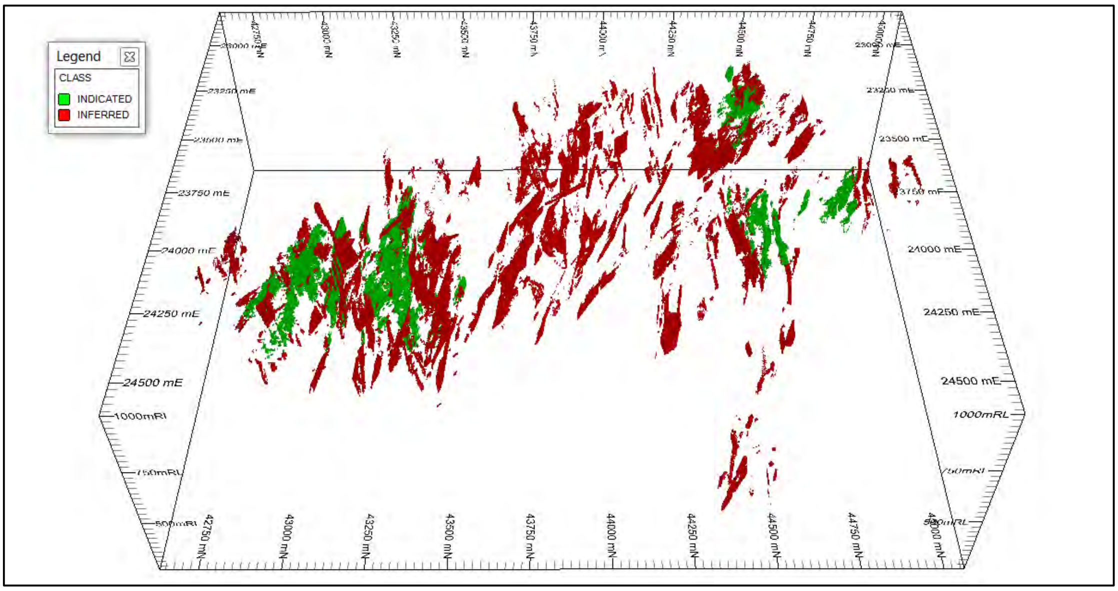


Figure 53. Classified block model showing Indicated and Inferred material, looking West at a 2.24 g/t Au Eq cut-off.

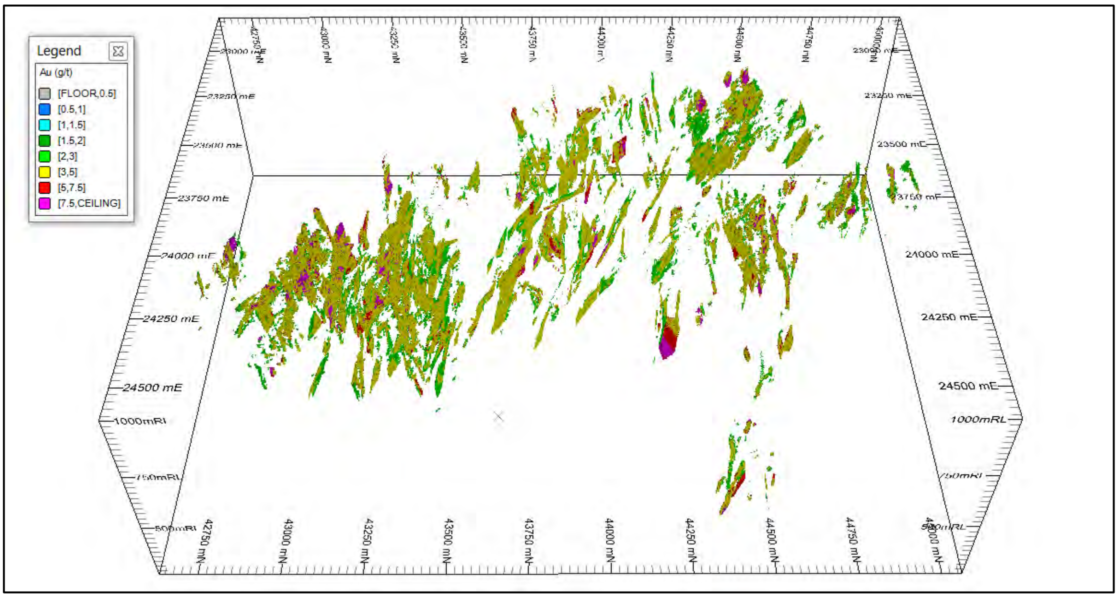


Figure 54. Block model coloured by Au, looking West at a 2.24 g/t AuEq cut-off.

14.8 Comparisons with previous MRE

The MRE has been compared with that previously reported for the resource by CSA in July 2013 in "NI 43-101 Technical Report Shahumyan Project, Kapan, Republic of Armenia" dated 29 July 2013 and filed on SEDAR. Key differences are as follows:

- All mineralisation wireframes used honour a minimum true width of 1.8 m
- A different AuEq equation was used to interpret the mineralisation and estimate grades.
- Underground channel sampling practises were verified and validated onsite and data from them were included in the current MRE (referred to as UC samples). UC samples were excluded from the previous MRE due to concerns over sample quality at the time of the site visit.
- A regression was used to estimate density from a well-defined relationship with AuEq grade, rather a single density average for all resources within the vein model.
- Mining depletion between 1 February 2013 and 31 December 2013.
- The current MRE is based entirely on modelled veins (wireframes). Approximately 34% of the tonnage and 28% of the Gold ounces from July 2013 MRE was estimated using probability methods.

The reported MRE differences at greater than a zero profit cut-off of grade 2.24 g/t AuEq are presented in Table 48.

The updated MRE has a 7% decrease in tonnes and a 3% increase in AuEq comprising 24% increase in Au grade, 4% increase in Ag grade, 7% increase in Cu grade and 26% decrease in Zn grade.

Of this, there was an 8% increase in Indicated tonnes, with a 3% increase in AuEq comprising 23% increase in Au grade, 16% decrease in Zn grade; 9% increase in Cu grade, and no change in Ag grade.

There was an 11% decrease in Inferred tonnes, with a 3% increase in AuEq comprising 23% increase in Au grade, 6% increase in Ag grade, 7% increase in Cu grade, 29% decrease in Cu grade.

The increases in Au and Cu grades were reviewed and are attributed to the use of UC samples in the December 2013 MRE. Exclusion of these grades in the previous MRE resulted in lower grades from outlying drill holes being extrapolated into blocks that are now informed by UC samples. Mining has confirmed the presence of these veins, and it was believed that the previous model understated grades in those areas.

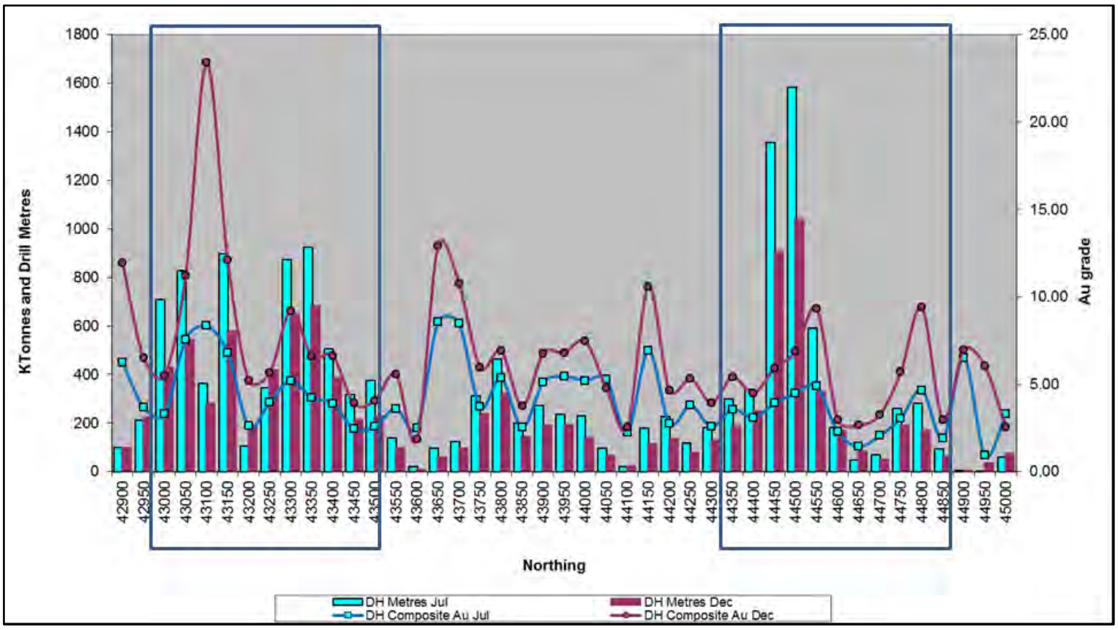
Zn grade has decreased and this has been attributed to better constraint of search neighbourhoods in the December 2013 MRE which has reduced a 'blow-out' at depth of higher Zn grades which previously had been extrapolated into poorly sampled areas. There may also be an impact from the use of diluted wireframes, which should increase the amount of low grade dilution into the estimate. However, if this is the case, it has only negatively affected Zn grades.

The decrease in Inferred tonnes is attributed to the conversion of Inferred material to Indicated. In addition, although large amounts of the areas previously estimated using probability methods have now been replaced by wireframed interpretations, there remains unwireframed mineralisation that previously had been modelled using probability. There may be an opportunity to wireframe some of this in the future, however, large parts of it are not well supported by existing drilling, and would benefit from infill drilling.

Figure 55 uses swaths to illustrate the comparison of July 2013 and December 2013 tonnage and grade at northing and bench increments, with areas dominated by UC samples highlighted.

Table 51. Table of comparison for December 2013 versus January 2013 MRE at a greater than zero profit and 2.24 g/t Au Eq cut off.

Dundee Precious Metals Kapan - Shahumyan Deposit Comparison 31 December 2013 versus 31 January 2013 Reported at greater than a profit cut-off of > USD 0 and a gold equivalent cut-off of 2.24g/t											
MRE	**Classification**	**Tonnes Mt**	**Au g/t**	**Contained Au Koz**	**Au Equiv g/t**	**Ag g/t**	**Cu %**	**Zn %**	**Pb %**	**S %**	**Density**
31 December 2013	Indicated	3.0	3.2	312	5.3	50	0.4	1.8	0.2	2.1	2.76
	Inferred	9.5	2.8	862	4.6	43	0.4	1.2	0.1	3.1	2.76
	Total	12.5	2.9	1,173	4.8	45	0.4	1.3	0.1	2.9	2.73
31 January 2013	Indicated	2.8	2.6	237	5.2	50	0.4	2.1	0.2	2.4	2.73
	Inferred	10.6	2.3	790	4.5	41	0.4	1.7	0.1	3.2	2.73
	Total	13.4	2.4	1,027	4.6	43	0.4	1.8	0.1	3.0	2.73
Comparison	Indicated	8%	23%	32%	3%	0%	9%	-16%	-6%	-13%	1%
	Inferred	-11%	23%	9%	3%	6%	7%	-29%	-1%	-2%	1%
	Total	-7%	24%	14%	3%	4%	7%	-26%	-3%	-4%	0%



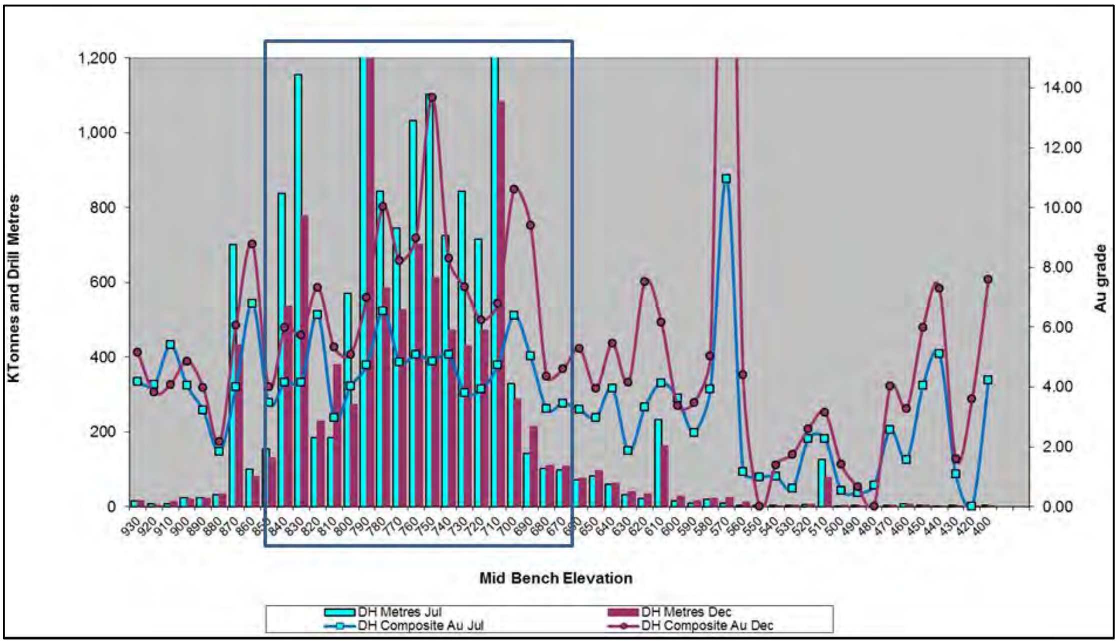


Figure 55. Swath plots by Northing (top) and Bench (bottom) of Au Eq showing Jul (blue) versus Dec (purple) total tonnages (bar) versus grades (line). Areas dominated by UC samples are enclosed in blocks.

15 Mineral Reserve Estimates

There are currently no defined Reserves for the Project.

16 Mining Methods

16.1 Introduction

DPMK's 100% owned Shahumyan underground mine is located adjacent to the town of Kapan in eastern Armenia. The current planned rate of production is 1,700 tonnes per day which is commensurate with the current milling rate of 600,000 tonnes per year.

However, for reasons discussed below, the current rate of mine production is around 1,000 tpd, in order to build up a higher level of drilled stope inventory. But in addition, it is planned to upgrade the underground mine to produce at a higher rate, with production ramping up in 2015 and 2016 so that by 2017, the mine is planned to produce at the rate of 1.0 Mtpa.

This upgrade envisages the introduction of additional new mobile equipment, the improvement of operations in the north section of the mine and a push on increasing the mine development rates. CSA believes that with the proposed technology, planned equipment purchases and the current Mineral Resource Estimates, this expansion is entirely feasible.

The reader is cautioned that this information is based on a PEA of the Shahumyan Project that is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and that there is no certainty that the PEA will be realised. No Mineral Reserves have been estimated. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

16.2 Mining

The underground mine has been in operation since the mid-20th century. However, the current mine has little similarity to the previous operations, although it does use some of the mine infrastructure such as the compressed air system and the mine ventilation system. Access into the mine is via two decline ramps, one to access the south section of the mine, and one to access the north section. Each decline has a portal at 780 mL (all levels are expressed as metres above sea level). Previous access into the mine was via vertical shafts. These shafts are still evident, but no longer provide any facility.

Crosscuts are developed from each of the declines to intersect the veins at approximately 90° in plan. Mine levels are at 10 m vertical intervals. Veins are near vertical. As each economic vein is intersected, a vein drive is developed along the vein to open up working faces. The capital development (declines and crosscuts) are developed using mechanised electric-hydraulic drill jumbos and vein drives are developed using hand-held pneumatic machinery.

To facilitate the movement of equipment around the mine, and to enable the introduction of larger equipment into the North section, a new connection is currently being built between the two parts of the mine. Completion of this connection is due in early 2015.

Sub-level open stopes are opened up along the vein drives. Each stope is typically 40 m high and 60 m along strike. Production blast holes are drilled upwards from the lower sub-level, and consist typically of 2-3 parallel holes. Current drilling is with Tamrock bar-mounted pneumatic rigs and a recently introduced Atlas Copco Simba drill jumbo. Stope holes are blasted into an initial stope raise and the stope is opened up from there. Rib pillars are left between stopes, and crown pillars are left between 40 m vertical stopes. These pillars are not presently recovered. On completion of each stoping area, i.e. an area of around 60 m on strike and 40 m in vertical height, the stope is available for filling with waste. All mine development waste is disposed of this way. The waste is not treated in any way, but does provide some support to the stope roof.

Broken material from the stope ring blasting falls to the mining drive where it is loaded by various types of load-haul-dump machines ("LHD") directly into diesel haul trucks and hauled to the surface. A list of the main mine mobile equipment is given in the following table. Remotely operated LHDs are in use in the fleet for final stope cleaning.

Table 52. Main mine production fleet

Model	Type of machine	Number in fleet	Capacity
Atlas CopcoST2G	LHD Loader	2	1.9 m3
Atlas Copco ST7	LHD Loader	1	3.2 m3
Atlas Copco ST710	LHD Loader	2	3.2 m3
Cat R1300G	LHD Loader	1	3.1 m3
Atlas Copco ST7	LHD loader	2	3.2 m3
MT2010	Haul truck	2	20 t
Atlas Copco MT426	Haul truck	1	26t
FAMT-12	Haul truck	2	8 t
Atlas Copco MT436B	Haul truck	2	32 t
Atlas Copco MT2010	Haul truck	1	20 t
Atlas Copco Boomer	Drill jumbo	2	Twin boom
Atlas Copco Simba 1250	Stope drill jumbo	1	120 m per shift

All current equipment is in a good operational condition, with the exception of the FAMT-12 haul trucks. These are considered unreliable and of poor construction. They will be retired within the next year. The mining methods in use in the north and south sections of the mine are similar, but capital development in the north section is done with hand held drilling equipment, whereas in the south section, all capital development is with mobile electro-hydraulic drill rigs. All mine vein development in both sections is done with hand held pneumatic machinery.

Stope drilling is done with Tamrock pneumatic rigs. The recently delivered Simba rig will replace up to three of the current pneumatic drills. It is expected to drill faster and more accurately than the pneumatic rigs, which will assist in the improvement of the stope blasting which in turn will lead to probable better dilution factors.

16.3 Mining Dilution

The veins containing most of the mineralisation are narrow, from 0.2 m in width and up to as much as 2.0 m in width, the average width being about 1 m. The mineral resource is modelled to a minimum width of 1.8 m which includes appropriate dilution. Due to geometric, geotechnical and operating issues, stopes can be wider than 1.8 m, resulting in further mining dilution. The amount of mining dilution is currently being analysed and procedures reviewed. The newly introduced drilling equipment is expected to improve efficiency and accuracy of the stope blasting and improve dilution factors.

16.4 Production Rates

At the time of the visit, the production rate from the mine was being constrained for the following reason. During the latter part of 2013, the inventory of drilled working areas was run down to such an extent that mine operations were being affected. Management took the decision that to ensure better continuity of extraction; a larger drilled stope inventory was to be maintained. It was proposed that a pre-drilled inventory of 150,000 t of mineralised material would be maintained, equivalent to a 3-month production inventory. Therefore, production is being held back until this inventory is achieved. It has also been proposed that a 6-month inventory of mine stope development is maintained as a minimum. Once this inventory has been built up, normal production will resume, and the Shahumyan LOM plan is proposed as presented in Table 53.

Table 53. Shahumyan Life of Mine Plan

LoM	Units	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Development Mineralised Material	000 t	1,372	177	187	173	214	192	178	166	65	20
Stope Mineralised Material	000 t	6,620	379	484	718	821	821	821	823	929	826
Total Mineralised Material	000 t	7,992	556	672	892	1,034	1,013	998	989	994	846
Mine Head grade Au	g/t	2.42	2.14	2.59	2.43	2.43	2.39	2.46	2.45	2.41	2.38
Mine Head grade Ag	g/t	37.8	38.5	39.9	37.3	35.8	35.1	34.3	36.9	35.8	46.6
Mine Head grade Cu	%	0.36	0.33	0.33	0.28	0.30	0.31	0.32	0.39	0.35	0.35
Mine Head grade Zn	%	1.05	1.32	1.32	1.16	1.13	1.04	0.85	0.88	0.78	0.95
Mine Head grade Pb	%	0.08	0.08	0.13	0.10	0.10	0.07	0.05	0.06	0.04	0.10
Mine Head grade S	%	2.39	2.72	1.96	1.79	1.96	1.90	2.42	2.55	3.12	3.12
Development Metres											
Hand held lateral	m	94,781	7,821	9,300	10,797	11,969	11,864	11,893	11,969	10,185	8,984
Jumbo lateral and decline	m	69,057	5,696	7,920	7,920	7,920	7,920	7,920	7,920	7,920	7,920
Total Waste	000 t	5,484	386	625	636	640	628	643	636	645	645
Total Material	000 t	13,476	942	1,297	1,528	1,674	1,640	1,641	1,624	1,639	1,491

16.5 Mine Development

All capital development drilling in the south section is done with the two Atlas Copco Boomer jumbos. All other development drilling in south section is done with hand-held pneumatic equipment. All development drilling in the north section, including capital development, is done with hand held equipment. When the new North – South access is completed, it is expected that all mine development will be carried out using jumbo drilling. DPMK reports that underground development is currently undertaken with the following dimensions; Capital development (North section) is 4 m x 4 m and in the South section (crosscuts) is 4.8 m x 4.8 m, (decline) is 4.8 m x 4.8 m. Eventually, mine development in the North section will be opened up to the larger size to enable the larger, more productive equipment currently in use in the South to be used all over the mine. Mine level development is standardised throughout both sections at 3.5 x 3.5 m. These dimensions are commensurate with the type of equipment in use and with the mining method in use.

The quantum of mine development over the life of mine is presented in table 50.

A single Boomer has the capacity to develop 150 m of capital development per month. Thus, it can be seen that capital development advance will remain constrained. It is expected that additional drilling capacity will need to be brought on line in both south and north sections. However, a programme to do this was not discussed during the site visit.

16.6 Mine Stability and General Conditions

No particular issues were noted during the site visit. Ground conditions underground were generally good. Some meshing is installed at major tunnel junctions. In the stopes, cable bolting is used to control the hanging wall of the stope in an effort to minimise dilution. It appears partially effective.

The stopes are filled with mine development waste after completion. This is more an effort to get rid of the waste material underground rather than to provide mine support, although this latter is a secondary result and can only add to the general stability of the area.

16.7 Mine Ventilation

The mine ventilation circuit is based on a main exhaust fan situated on surface. This fan, powered by an 800 kW electric motor, is exhausting around 160 m^3/second of air.

Ambient conditions underground were good, even where diesel equipment was working. Underground air control is with ventilation doors and booster fans.

16.8 Underground Water and Pumping

The mine cannot be described as a wet mine. What water does enter either the North section or the South section is drained to a sump on the South side of the mine at the 703 m elevation. A single Flygt 37 kW electric pump is installed which operates as and when required. A second, 20 kW Flygt pump is available as a standby. Actual pumping figures were not available but the volume of water is not significant. More important is that fact that any sludge is handled by the Flygt pump, and this sludge, which contains metal values, is mixed with the ROM feed to the plant.

A new pump station has been excavated on the 680 m level, to ensure continued pumping as the mine develops deeper. This will be equipped with Mono pump, which are in site awaiting installations.

16.9 Underground Refuge Station

A number of underground refuge stations have been constructed, each in a central position near the main decline. Each consists of a container type of unit which can be sealed when in use. A number of services are installed, such as a video and voice link with surface, oxygen bottles, compressed air, emergency supplies and first aid supplies. It will provide a safe haven in the event of any event that might affect the environment underground, such as the release of poisonous gases from a fire. Two refuge stations are installed in the North section and two in the South section.

17 Recovery Methods

The sections below summarise aspects of the processing plant, design, equipment and specifications. Summary information relating to requirements for water and power is contained in Section 18.

17.1 General Description of the Process Plant

Run-of-mine material (ROM) from the north section is hauled out of the north decline, and transported and tipped by trucks onto the main stockpile area. The ROM from the south section is hauled and tipped in the stockpile area, again by truck. Material from either section is rehandled with a Cat 996 loader into the feed bunker during the crushing stage, while managing any oversize rocks for size reduction by a mobile rockbreaker as required.

The primary crusher is a 1200 mm x 900 mm jaw, with the unit discharge (at -150 mm) feeding directly into the secondary cone crusher for reduction to 75 mm; this product is the direct feed into the KMDT 2200 tertiary cone crusher for a reduction to a nominal -25 mm. The crushed product is the feed to the grinding circuit, and is conveyed to one of two storage bunkers, each of approximately a 1,000 t storage capacity.

ROM from the bunkers is fed by conveyor to the primary mill (rod mill - 2.7m dia, by 3.6m long, 400 kW) in the plant. Oversize from the mill is fed to a second stage ball mill as the sands product from within in a closed circuit with a spiral classifier. Classifier overflow is fed to the copper flash flotation circuit. Tails from the flash flotation are passed through a second stage hydrocyclone classification with tertiary grinding through a ball mill in a closed circuit. Hydrocyclone overflow is feed to the copper flotation circuit, comprising of 2 rougher and 2 scavenger cells. The rougher concentrate is cleaned in the 3 stage cleaner circuit, while the scavenger concentrate is combined with the first cleaner tails and returns to the head of the roughers.

Copper scavenger tails is the feed to the zinc circuit, comprising of 6 rougher and 8 scavenger cells. The rougher concentrate is cleaned in the 3 stage cleaner circuit while the scavenger concentrate is combined with the first cleaner tails and returns to the head of the roughers.

All of the float cells in the copper and zinc circuits are relatively new, with some dating back to 2005 and the newest being installed in the copper circuit in 2012.

The current operating strategy of plant operation while the mine is ramping up its development rates is to build a stockpile and process the material in batches at a nominal rate of around 70 t/h (1,700 t/d). This is well below the maximum recorded capacity of one line (~2,100 t/d), while the nominal capacity of the flotation section is now well above this following the additional capacity installed in 2012.

Copper recovery to the final copper concentrate is 85%, with a target recovery of 90% based on continuous improvement of the circuit. Gold recovery to copper concentrate varies between 68 to 74%, with the grade of gold in the copper concentrate at 90 to 110 grams per tonne (g/t). The grade of the copper concentrate is typically of 20 – 22% copper, while that of the zinc concentrate is typically 58 - 63% zinc. The zinc concentrate also contains a further 10 to 15% of gold distribution, resulting in an overall site gold recovery of 80 to 85%.

Concentrates are thickened by 24 m thickeners and filtered by drum filters to produce a final dry concentrate. Normal moisture content is around 7%. In some cases, a flocculant is used in the concentrate thickeners, to speed settlement of the fine material. When a flocculant is used, moisture content in the concentrate increases to around 11% which is close to the limit for bulk shipment.

Copper concentrate, shipped in bulk bags, is transported by truck to Yerevan, and shipped onwards by rail to the port of Poti in neighbouring Georgia. Zinc concentrates follow the same route but are shipped in bulk in containers.

As part of the upgrade to 1 million tonne capacity, DPMK plans to refurbish the original second mill line, comprising of 3 mills (one rod and two ball), a spiral classifier and crushed material bunker. This will also include replacing the complete mill feed system, the installation of a hydrocyclone battery, as well as several new flotation units to upgrade the capacity. The upgrade plan will effectively double the current grinding circuit capacity and will introduce the additional flotation capacity needed to process the increased throughput. The estimated cost of the upgrade program for the mill is $ 6 million as further detailed in Section 21. Another $13.4 million will be required to upgrade the existing tailings management facility to handle the increased volume of process tailings. Table 51 presents the predicted balance of concentrate/tailings for the LOM after the upgrade is completed in 2016.

Table 54. Predicted LOM Metallurgical Balance (2016-2024)

Life of Mine	%Wt	Cu , %	Au , %	Zn , %	Ag , %
Cu Conc.	1.5	89.0	73.0	5.0	73.0
Zn Conc.	1.5	7.0	7.9	89.0	9.1
Tails	97.1	4.0	19.1	6.0	17.9
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

17.2 Tailings Management Facility

Final tailings are pumped to the current tailings management facility (“TMF”) at Geghanush. This is located in a valley and has a north and south wall. The north wall was raised in April 2012, and together with the south wall, previously raised, the TMF has a life of 10 years with a throughput of 600,000 tpa with the next raise planned within one year. In order to increase the throughput of the Process Plant to 1Mtpa, further, the upstream extension of the TMF is required, which is described later in this report.

Piezometers are located around the TMF to enable ground water sampling to take place.

A water decant reclaims water for the mill operation. Current water usage in the mill consists of 70% reclaim water from the TMF and 30% fresh make up water.

The TMF is particularly well managed and maintained. The previous TMF, the Artsvanik dam, is now closed, and has been rehabilitated by Dundee. The old TMF has been re-contoured and re-vegetated and the appearance of this work is particularly good. It has been returned to pasture. A concrete water flume has been constructed around the old facility to avoid scouring and erosion by rainfall.

18 Project Infrastructure

18.1 Electrical Power Supply

Power used on site is supplied by the state–owned power corporation. Power at 33 kV is transformed at Shahumyan to 6 kV for site distribution. On-site facilities, including the mine and plant have their own transformer stations for district reticulation at 1,000 V and 400 V, respectively.

Power is stated not to be a constraint at current mine production rates. The expansion capital cost estimate includes upgrades of the main distribution substation 35/6 kV, switchyard and indoor distribution board 6 kV.

18.2 Process Water

Much of the water for the plant and mine is recirculated from the TMF. Make up water is taken from the Voghdji River that flows through the town. There is no evidence of water shortages.

18.3 Compressor Station

Compressed air is used underground for drilling. A compressor station, dating from Soviet times, is still maintained at the site. This is a substantial building housing four Soviet era reciprocating compressors. One is rated at 120 L/s FAD capacity, and is powered by an 800 kW electrical motor. The remaining three are rated at 100 L/s FAD and is each powered by a 630 kW motor. All deliver air at 8 bar pressure. Normally, one unit is in use with a 3 on standby.

Although around 50 years old, these units are kept in excellent condition and have performed well over the years. There are no plans to replace them.

18.4 Transport Facilities

The town of Kapan is on the national road network and is reached by standard vehicles. The site elevation is around 800 m above sea level. There is no direct connection to the national railroad system.

There is a disused airfield in town, with a paved runway around 1,700 m in length. This reportedly could take Yak-40 jets and An-24 type turboprop aircraft. It appears useable in an emergency.

19 Market Studies and Contracts

19.1 Markets

For the 2014/15 metal production, the copper and zinc concentrates are being sold to several metal traders and a smelter. Concentrates are trucked to the Armenian capital Yerevan with the copper concentrate being transported in bags (2 tonne), and zinc concentrate in bulk. Loads are transferred to rail transport in Yerevan and currently transported to Poti port in Georgia on the Black Sea. At the time of the technical visit, and for the foreseeable future, the copper concentrates were being routed to Chinese copper smelters and zinc concentrates to a European smelter.

19.2 Contracts

The terms of smelting, refining, transportation, handling, sales, hedging, forward sales, contractor arrangements, rates or charges, are within market parameters for the type of Copper and Zinc concentrates that DPMK produces. The company does not use mining or concentrating contractors as the mining and mineral processing activities are self-performed.

20 Environmental Studies, Permitting, and Social & Community Impact

20.1 Environmental Studies

DPMK has been conducting comprehensive baseline environmental studies over the Kapan licence area since 2007. The studies were carried out by its own personnel as well as a combined group of local and international companies - The Mining and Metallurgical Institute (Yerevan, Armenia), Golder and Associates (Canada), Kingett Mitchell (New Zealand), AATA International (USA), and Geolink Consulting (Russia).

The studies included:

- Environmental, climate, socio- economic and community study baseline studies
- Air quality and noise monitoring,
- Hydrological and ground water studies, water quality sampling,
- Aquatic ecosystems (including aquatic, benthic, periphyton and fish invertebrates) studies,
- Terrestrial flora and fauna studies, and
- Cultural history and archaeology.

The baseline study concluded that while mining operations did have an impact on various of the measured parameters, impacts on the local communities were non material.

Following the baseline study, a significant monitoring program was implemented and continues. Regular (weekly and monthly) reports are provided internally, with quarterly and annual reports being provided to the relevant (regional and federal) state authorities, and to the DPM Corporate Group

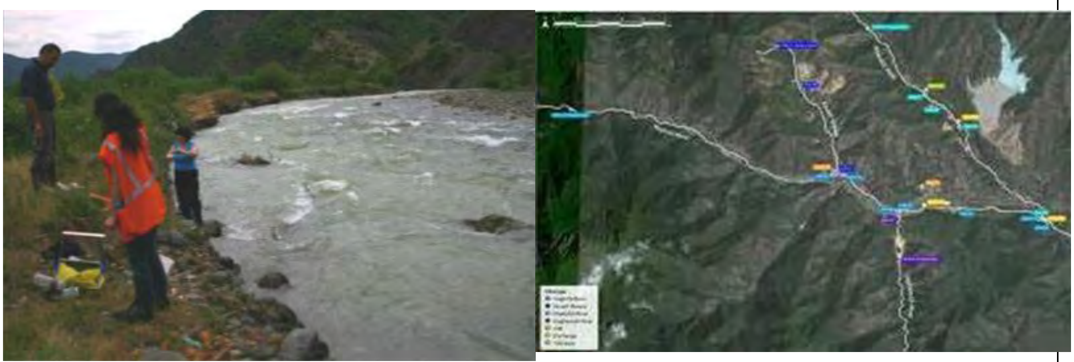


Figure 56. River sampling station (left) and surface water quality monitoring sites (right), (Deno Gold 2011).

The site is a legacy site from Soviet times. It is an intrinsic part of life in the town. However, there are no issues that can be seen as negative. The mine is on the edge of the town and hardly impinges on the daily life of residents. There are no noxious discharges, and little noise pollution.

The operation currently recycles 100% of the TMF decant water and 75% of mine water. Fresh water make-up is extracted from the local river. At current production rates, the operation is working well within the permitted volumes, and this is expected to be the case during the expansion. The remaining 25% of water coming from the mine is being discharged into a river as per the current water intake and discharge permit. All waste from underground is backfilled into the old stopes and no new waste dumps are being formed. There are legacy waste dumps from old exploration adits. The quality of discharges from these dumps was not completed during the base-line study, however is currently being properly evaluated. The full implications from a legal requirement for closure remains to be confirmed with the appropriate Ministry, which would be part of any future Operating Permit negotiations.

DPMK currently has no liability for mining of the Centralni open pit mine (closed prior to take over in 2004). A conservation plan was developed for the mine which was further approved by the relevant state authorities. Activities proposed by the conservation plan were successfully completed. The license was returned back the Government of Armenia along with the remained liabilities.

No significant environmental liability, other than closure of the current operation, is currently held by the company.

20.2 Local Community Issues

DPMK has positioned itself as an integral part of the local Kapan community. Currently all interactions with communities are planned, and routine projects and/or operations related. Specifically, there are four direct stakeholder communities, three rural and one urban, in which the company is particularly active. Project related social programs are also performed and are included in the initial project development documentation and may vary in their nature depending on the project stakeholders' requirements.

Social programs continue to be addressed to different vulnerable groups existing in the surrounding communities. These programs mainly address improvement of children daycare facilities; donations for health care to vulnerable groups; infrastructure improvement of direct stakeholder communities; environmental conditions improvement activities, amoungst others.

Details of the various activities are included where relevant in the annual DPM Sustainability Reports (2012, 2013), which covers all of the company operations.

20.3 Tailings Management Facility (TMF)

Tailings from the mill flotation circuit at DPMK are handled at an existing tailings management facility (TMF) situated in the Geghanush Valley. In year 2012/2013 the options for the construction of the northern TMF and the closure of the Geghanush TMF were studied and two reports were prepared:

- Geghanush Tailings Management Facility Closure Study (Golder Report No. 12514150374.501/B.0).
- Northern Tailings Management Facility Pre-Feasibility Study (Golder Report No. 12514150374.502/B.0).

Due to the high capital cost in year 1, an alternative to extend the life of Geghanush was reassessed by Golder in the following memorandum;

- Geghanush Tailings Management Facility Life Extension (Golder Technical Memo No. 14514150045_TM2/16.06.2014).
- As a result of this later work the Geghanush TMF life extension project has been accepted as the preferred option.

This section summarises the pertinent points discussed in the latter report with regards to the concept for extension and closure.

The estimates are based on an increase in the mine production throughput from 0.55 Mtpa to 1.0 Mtpa in Year 3.

20.3.1 Geghanush TMF

20.3.1.1 Current Operation

The TMF is constructed in a river gulley, and includes a southern, downstream embankment and a northern, upstream embankment. The Geghanush TMF is currently in use today with an upstream extension. The TMF was re-commissioned in 2006 following the design and implementation of a culvert to divert the Geghanush River. Although the TMF still has free capacity, an upgrade is required to provide for the next 8 years of operation.

20.3.1.2 Design and Operation

The conceptual design entails raising the existing TMF and extending the facility upstream to the south. Both the northern and southern embankments will be raised through the years by 21 m using the upstream method. An additional embankment will be commissioned upstream of the existing facility. The Geghanush River will be diverted into a new concrete channel on the east side of the valley around the TMF and its new southern extension.

Table 55 presents the tailings production rates for current and ramp-up throughput.

Table 55. Tailings Production Rates

ITEM	Units	Current Rate 0.55Mtpa	Expansion Rate to 1.0Mtpa
Annual Tailings Production	Mtpa	0.55	1.0
Tailings Dry Density	g/m^3	1.6	1.6
Annual Tailings Production	Mm^3/year	0.344	0.625
Total Production	Mm^3	5.16	5.15
Design Life	Years	15	9

The facility will have a storage capacity of 5.1 Mm^3 and a closure elevation of 833 m asl and 830 m asl for the extension. For a production rate of 0.9 Mtpa, the facility will have storage capacity for 9 years.

20.3.1.3 Conceptual Closure Design

Tailings from the mine are expected to be deposited up to the final elevation of 831 masl. This will give a surface area of approximately 34.1 Ha. The closure will entail reprofiling the basins, and once the tailings have consolidated to a sufficient degree to support earthmoving equipment, the facility is to be capped with a 250 mm layer of inert soils and topsoil to re-establish vegetation, with an irrigation system to be commissioned to support this.

20.3.2 Cost Estimates for Closure

The closure cost estimates are based on the Technical Memo prepared by Golder Associates (Golder Associates, Report 1), and are based on standard Closure Cost assumptions, including - the use of local materials and supplies; local construction rates by suitably qualified contractors (or rates from other equivalent projects); supervision and general costs as standard percentages of the the complete project; combined with suitable allowances for monitoring and maintenance of the facility after closure.

Cost estimates for closure (±30%) are presented in Table 56.

Table 56. Cost Estimate for closure of the existing Geghanush TMF

ITEM	Amount (USD,000)
Allowance for erosion control measures for closure	1,200
Topsoil to the TMF for closure (minimum 250mm thick)	572
Allowance for cost of re-vegetation	362
Allowance for TMF maintenance post closure	400
TOTAL COSTS (USD)	**2,534**

20.4 Facilities Closure

Closure costs for the operating facilities have been broadly estimated based on the areas of the facilities of the mine and plant areas, while those for the TMF are given in Table 56. The (+/-30%) estimate for closure of all the facilities is M$ 12, with M$ 2.6 for the Geghanush TMF.

21 Capital and Operating Costs

21.1 Capital Costs

The reader is cautioned that this information is based on a PEA of the Shahumyan Project that is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and that there is no certainty that the PEA will be realised. No Mineral Reserves have been estimated. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

21.1.1 Mine fleet initial growth capital

An initial growth fleet is proposed to boost the mine production up to an annualised 1.0 Mtpa. The number of units is based on current productivity and experience in the conditions prevailing at Kapan. Prices used are based on budget quotations received. This information is presented in Table 57.

Table 57. Mining Mobile Capital for Initial growth

Mine - growth capital	Cost per unit $000	New fleet	Total fleet cost $000	Year 1	Year 2	Year 3
USD ('000)						
Jumbo	700	2	1,400		700	700
Simba	630	1	630		630	
Big Loaders	625	2	1,250			1,250
Trucks- Small	635	0				
Trucks - Big	855	5	4,275		2,565	1,710
Support Equipment			2,000		1,000	1,000
Other Underground Equipment Capital			1,000		500	500
Total U/G Equipment			**10,555**	**0**	**5,395**	**5,160**

21.1.1 Mine development and construction

Mine development and construction costs are presented in Table 58 and represent both growth capital and sustaining capital. The costs are only the jumbo based capital development costs and are based on current development costs. Hand held development costs are included in operating costs.

Table 58. Mine Capital Development Costs

Costs in $000	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Mine capital development	9,918	13,789	13,789	13,789	13,789	13,789	13,789	13,789	13,790

21.1.2 Process plant capital cost

The plant currently has a throughput capacity well in excess of the planned 0.6Mtpa of ROM feed. To achieve the planned throughput increase to 1Mtpa, the existing (but not utilized) second grinding line will be refurbished and brought back into operation. The capital costs for this, together with the other upgrades required within the current mill and outside facilities (including the TMF) have been estimated following a detailed study completed (DPM Projects Group, Report 20140825).

The summary of the plant expansion capital cost, including the expansion of the tailings dam and also for other surface equipment is presented in Table 59.

Table 59. Capital Cost – Process Plant

Mill	USD ('000)			
	Year 2	Year 3	Year 4	**TOTAL**
Expansion from 600k to 1Mtpa - Equip & Install		6,134		**6,134**
Tailings Dam Expansion	300	6,720	6,420	**13,440**
Other Surface Equipment Capital				
Total	**300**	**12,854**	**6,420**	**19,574**

Further details of estimated capital and sustaining expenditures are presented in Tables 59, 60 and 61.

21.1.3 Capital Cost – TMF Upgrade

The cost estimates are based on the conceptual designs for the TMF and storm water management system as part of the study undertaken by Golder Associates (Golder Associates, Report 2). Capital and Operating Costs were developed based on an annual basis for the development of the facilities.. The cost estimates are based on the following assumptions:

- The TMF embankments will be constructed using material from the mine site as far as possible, providing that the material will be suitable for the construction.
- Locally available materials will be used as far as possible for construction activities to reduce the cost of the works.

- Where additional material is required for embankment raises, the material will be imported.
- The works will be constructed by suitably qualified contractors.
- Allowances have been included for piezometers and for the installation of survey control beacons.
- Allowances have been made for construction works required to extend the TMF (capex for the initial construction and opex for operating the facility) as well as to rehabilitate the TMF (as part of closure) and prepare the facility for closure.
- An allowance has been made for re-vegetation upon closure and for the monitoring and maintenance thereof during the post closure period (as part of the closure).
- As far as possible rates have been based on local construction rates received from DPMK and civil and earthwork contractors in Armenia. When local rates were not available, Golder Associates also used rates for similar projects carried out over the past few years in Ireland.
- The opex for the facility for future wall raises is also included

21.1.3.1 Operating Costs

The opex for the TMF includes the expansion of the facility, as well as subsequent embankment raises, access roads and the associated infrastructure (including the storm water management system and measures), required during the Life of Mine. Closure costs have also been included as part of the opex costs.

21.1.3.2 Capex and Opex Summary

A summary of the capex and opex for the TMF and the associated infrastructure (including Tailings Pumping Systems), including the cost per tonne for tailings disposal over a 9 year operational life is presented in Table 60. The cost estimates are at an accuracy of ±25%.

Table 60. Upgraded TMF, 9 Year Life – 0.9 Mtpa

ITEM	Costs for selected TMF Option USD ('000)
Capex	13,441
Sustaining capital for wall raises	457
Opex Man Power and Pumping	2,978
Closure	2,965
Total	**19,841**
Tailings Tonnage Capacity (9 Years)	8.2Mt
Cost/tonne (USD/t)	2.41

21.1.4 Sustaining capital

In addition to the initial capital in years 3 and 4, there will be on going sustaining capital which is intended to maintain the levels of production at the stated rate, and which will be used for replacement of obsolete equipment, mine capital development, buildings, licences, permits, and geological drilling for resource expansion and upgrade.

Table 61. Sustaining Capital - Mining

USD ('000)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	TOTALS
Mine fleet replacement											
Jumbo			1,400			0	700				2,100
Simba					630	0	630				1,260
Big Loaders	1,792	3,750	1,250		625	1,792	0				9,208
Trucks- Small	624	1,270				624					2,518
Trucks - Big	2,246	855	855			2,246	2,565				8,767
Support Equipment							1,000				1,000
Total U/G Equipment	**4,662**	**5,875**	**3,505**	**0**	**1,255**	**4,662**	**4,895**	**0**	**0**	**0**	**24,854**

Table 62. Sustaining Capital – Process and General

USD ('000)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	TOTALS
Plant and surface general												
Plant and Equipment Capital	1,980	1,000	1,000	2,000	2,000	2,000	2,000	1,000	0	0	0	12,980
Mobile Equipment Capital	773	600	600	600	600	600	600	600	600	0	0	5,573
Land, License, and Permit Capital	0	500	500	0	0	0	0	0	0	0	0	1,000
Buildings or Other Infrastructure Capital	1,439	100	0	150	100	0	150	100	0	0	0	2,039
Resource Development Diamond Drilling	1,230	1,070	1,442	1,677	1,625	1,955	1,968	1,364	0	0	0	12,330
TMF wall raise	51	51	51	51	51	51	51	51	51	0	0	459
Other including environmental costs	517	660	760	720	620	710	570	570	470	340	200	6,137
Total Plant and surface Sustaining Capital	**5,989**	**3,981**	**4,353**	**5,198**	**4,996**	**5,316**	**5,339**	**3,685**	**1,121**	**1,090**	**200**	**40,517**

Table 63. Capital Cost Summary

USD ('000)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	TOTALS
Mine expansion	0	5,395	5,160	0	0	0	0	0	0	0	0	**10,555**
Mill expansion	0	0	6,134	0	0	0	0	0	0	0	0	**6,134**
TMF expansion	0	300	6,720	6,420	0	0	0	0	0	0	0	**13,440**
Other growth capital	1,878	0	0	0	0	0	0	0	0	0	0	**1,878**
Mine capital development	9,918	13,789	13,789	13,789	13,789	13,789	13,789	13,789	13,790	0	0	**120,234**
Mine fleet replacement	4,662	5,875	3,505	0	1,255	4,622	4,895	0	0	0	0	**24,854**
Plant and surface Sustaining capital	5,989	3,981	4,353	5,198	4,996	5,316	5,339	3,685	1,121	340	200	**40,517**
Closure Capital	0	0	0	0	0	0	0	50	2,500	5,000	4,455	**11,995**
TOTAL	**22,446**	**29,339**	**39,661**	**25,408**	**20,041**	**23,767**	**24,023**	**17,524**	**17,411**	**5,340**	**4,645**	**229,606**

21.1.5 Operating Costs

The current overall site operating cost, excluding G&A, is USD 68.00 per tonne of processed material. As production is increased with the throughput ramp-up, significant efficiencies of scale should be realized, as presented in Table 62. The average operating cost, excluding G&A, per tonne of mineralised material over the mine life is expected to be USD 51.66

Table 64. Summary of operating costs

USD/t	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Mining	26.81	21.36	18.82	17.99	18.55	19.06	19.54	19.90	20.37
Process	17.95	17.72	16.97	16.66	16.70	16.73	16.75	16.74	16.76
Services	17.83	9.69	7.30	6.30	6.43	6.52	6.59	6.55	6.61
Royalty	5.41	11.53	10.57	10.37	9.23	8.29	8.18	6.19	5.28
TOTAL	**68.00**	**60.31**	**53.67**	**51.31**	**50.91**	**50.61**	**51.06**	**49.38**	**49.02**

22 Economic Analysis

DPMK has completed a Preliminary Economic Analysis based on the capital and operating costs from Section 21 of this report.

The reader is cautioned that this information is based on a PEA of the Shahumyan Project that is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and that there is no certainty that the PEA will be realised. No Mineral Reserves have been estimated. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The estimated economics for the Kapan operation is as presented in Table 65. The metal prices used are presented in Table 65 for each year. CSA approves of the inputs into the cash flow analysis, and state that it provides a reasonable assessment of the value of the project.

No inflation factors have been used in the analysis.

The summary results of the base case generate an NPV (at 5.0% discount rate) of USD 141.7.

LOM Project capital (excluding sustaining capital and closure costs) is estimated at USD 30.1M.

Table 65. Summary Economics

Project Results Summary (LOM)		
Item	**Unit**	**LOM**
Throughput	Mtpa	0.9
Project Life	Years	9
Gold Price	USD/oz	1,300
Copper	USD/lb	3.00
Silver Price	USD/oz	20
Zinc	USD/lb	1.00
Production and Revenue Summary (LOM)(1)		
Item	**Unit**	**LOM**
Total quantity of material mined/milled	Mt	7.6
Gold grade	g/t	2.44
Silver grade	g/t	37.60
Copper grade	%	0.33
Zinc grade	%	1.00
Overall metallurgical recoveries		

Gold recovery	%	80.9
Silver recovery	%	82.1
Copper recovery	%	96.0
Zinc recovery	%	94.0
Total Net Revenue	USD M	874.7
Site EBITDA	USD M	417.1
Average annual payable production		
Gold	ozs	54,084
Silver	ozs	824,757
Copper	M lbs	5.8
Zinc	M lbs	15.8
Operating and Capital Cost Summary (LOM)		
Item	Unit	LOM
Throughput	Mtpa	0.9
mining cash cost per tonne of material	USD/t	19.36
processing cash cost per tonne of material processed	USD/t	16.84
Services cash cost per tonne material	USD/t	6.88
royalty per tonne of material processed	USD/t	8.57
admin cash cost per tonne processed	USD/t	8.74
Total cash cost per tonne of material processed	USD/t	60.40
Cash cost per ounce of gold sold, net of by-product credits(2)	USD/oz	336
Capital Costs		
Initial capital	USD M	30.1
Sustaining capital	USD M	165.0
Closure and rehabilitation costs	USD M	12.0
Total LOM capital	**USD M**	**207.2**
Project Economics		
Average Annual EBITDA	USD M	52.1
NPV at a discount rate of 5.0%	**USD M**	**141.7**

(1) *Excludes first year from life of mine schedule which is deemed to have occurred in 2014*

(2) *Cash cost of sales per ounce of gold sold, net of by-product credits, represents cost of sales, less depreciation, amortization and other non-cash expenses, plus treatment charges, penalties, transportation and other selling costs, less by-product zinc, copper and silver revenues, including realized gains on copper derivative contracts, divided by the payable gold in concentrate sold.*

22.1 Sensitivity Analysis

Sensitivity analysis has been conducted to assess the effects of changes in key parameters upon the NPV. The analysis encompasses the range of +/-10%, 20% and 30% of the following key parameters:

- Gold price
- Copper Price
- Gold recovery
- Copper Recovery
- Aggregate operating costs
- Capital costs

In assessing the sensitivity of the Project returns, each of these parameters is varied independently of the others. Combined beneficial or adverse variations in any of these parameters will therefore have a more marked effect on the economics of the project than the individual variations considered.

Table 66. Sensitivity Analysis

	NPV (5.0%)						
Factor	**(30.0%)**	**(20.0%)**	**(10.0%)**	**–**	**10.0%**	**20.0%**	**30.0%**
Gold Price	$44	$78	$110	$142	$172	$203	$233
Copper Price	$118	$126	$134	$142	$149	$157	$164
Gold Recovery	$45	$78	$110	$142	$172	$202	$217
Copper Recovery	$126	$131	$137	$142	$147	$152	$157
Operating Costs	$209	$187	$164	$142	$119	$95	$71
Capital Costs	$193	$176	$159	$142	$125	$107	$90

23 Adjacent Properties

The Shahumyan deposit is located within a significant metallogenic province which hosts some of the largest deposits of the Caucasus region, however there are no local “adjacent” Properties currently in operation.

24 Other Relevant Data and Information

24.1 The Potential for "Pre-Concentration" at DPMK

The operation currently produces separate copper and zinc concentrates by a sequential flotation route. The bulk of the unrecovered components are mainly 'misplaced' metals into either of the other concentrates – a common feature of sequential base metal flotation circuits, with the final tailings containing relatively low levels of the economic metals. The overall plant operational results have been presented in Section 13, which show a consistent improvement in performance overall since the acquisition by DPM in 2007.

As part of DPMK's continuous improvement process, alternative processing options have been examined to determine if, and if so, how, additional improvements in Metallurgical processes can be incorporated. As outlined in Section 13, the basic mineralogical characteristics of various samples have been reviewed, and several feed materials have been tested to determine if "pre-concentration" by gravity ("heavy or "dense" media) processes would be possible. For this, a typical mill feed sample, and a very low grade (waste) sample were subjected to a standard laboratory density separation procedure (SGS 2013) to determine the component mineral liberation characteristics.

Table 67 presents the potential weight and corresponding metal losses to the respective reject portion at 60 and 80 per cent weight rejections – these correspond to fluid density levels of between 2.72 and 2.78 g/cm^3.

Table 67. Potential Weight Rejection v.s Metal Loss (Dec 2012 Sample)

	Low Grade		Rod Mill Feed	
% Weight Rejected	60	80	60	80
% Metal Rejected - Cu	9	12	11	16
Au	10	14	12	18
Ag	7	10	6	11
Zn	8	10	8	11

The 'low Grade' sample taken from the ROM stockpile (December 2012) did manage to upgrade to typical mill feed grades for the "pre-mid 2013 mineralisation/waste mining strategy" where significant dilution was normal procedure (see Section 16 for details on the current procedures for minimising dilution). The rod mill feed sample, taken from the stockpile in February 2013 feed material also demonstrated moderate metal losses with a high weight rejection, as presented in Table 68.

Table 68. Potential Weight Rejection vs. Metal Loss (Feb 2013 Sample)

"Low Grade Sample"	%Wt Reject	Au	Ag	Cu	Zn
DMS Feed Grade	0	0.33	6.7	0.06	0.47
Plant Feed Grade	78	1.32	25.1	0.24	1.97
Plant Feed Grade	90	2.64	51.0	0.49	3.90
% Metal Loss	78	12	8	11	8
% Metal Loss	90	20	15	18	17
"Rod Mill Feed Sample"	%Wt Reject	Au	Ag	Cu	Zn
DMS Feed Grade	0	1.51	40.3	0.28	2.09
Plant Feed Grade	60	3.32	65.8	0.62	4.81
% Metal Loss	60	12	8	11	8

The implications for the operation are significant, and while the figures quoted are for illustrative purposes to show the potential benefits of installing a “Pre-Concentration” stage at Kapan, more testing is being undertaken to a more detailed level to confirm these findings. The work will properly quantify the potential benefits, and confirm the ranges of these preliminary findings.

24.2 Implementation at DPMK

As part of the PEA processing testwork program, the requirements for the production of a “pre-concentration” circuit was examined to determine the optimum location to install such a unit within the current infrastructure.

At the full mine production rate of 1 Mtpa, approximately 50% of the overall feed material could be upgraded as the mill feed after primary and secondary crushing to the current F_{80} size of 25 mm. The remaining 50% would be a ‘waste’ product which would be transported by truck or conveyor as required.

Pre-Concentration Plants are common in certain industries, and are commonly provided as pre-engineered stand-alone packages[1]. This is the approach currently being considered as being particularly applicable for DPMK, and various options of locating the unit have been considered. The installation of such a circuit would fit into the current infrastructure, and once the increased mining rate is achieved, the mineralised material would continue to be delivered to the current primary crushing circuits, which will need some upgrading. The existing milling, flotation and tailings systems will be used.

[1] Particularly in Diamond Processing Plants.

24.3 Metallurgical Testwork

Further flow sheet test work and conceptual studies will be conducted in the next three months to complete the basis for input to the feasibility study.

24.4 Project Infrastructure

Final circuit definition, including the engineering and equipment sizing and selection, will be confirmed based on the ongoing characterisation and pilot plant results which will be incorporated into a separate Feasibility Study.

24.5 Permitting

The Project is likely to bring additional obligations related to closure and rehabilitation of the new facility, and more particularly in the areas of storage of the waste. These issues will be fully addressed during the next stage of the project development.

24.6 Project Implementation Stages

The Pre-Concentration Project is expected to be implemented in stages. The first stage to define the potential operating envelope based on a range of actual and future material feeds expected. This will be completed by examining the variability of current samples being subjected to the pre-concentration testing at a recognised testing laboratory, followed by modelling and pilot testing to produce bulk samples for process characterisation and confirmation.

The second stage would be the construction and installation of the Package, which would then occur prior to any substantial plant capacity upgrades that would be necessitated by the increase in mine production to 1 Mtpa.

24.7 Conclusions

The test program completed to date has generally confirmed the process proposed for the Kapan expansion will be relatively easily incorporated as an “add-on” facility in conjunction with the infrastructure presently incorporated in Kapan. This will require a separate, and new package facility to ‘preconcentrate’ the mined mineralised material by rejecting a product of sufficiently low metal content to be defined and classified as waste material. This is expected to be a minimum of 50% of the feed weight. The remaining product will be processed in the existing concentrator, at no greater than current throughput rates, but at significantly enhanced feed grades compared to those currently being processed.

This will effectively significantly reduce the processing operating costs on a per tonne of feed basis by >40%.

24.8 Recommendations

- The current 'preconcentration' project development has demonstrated sufficient financial upside to be more fully examined at the next level of study, and this recommendation has been made to DPM senior management for the funds to complete this level of study over the next three to 12 months. This includes (and is not limited to):

    - Work will need to be completed in all areas of project development to confirm the process assumptions for each of the stages proposed – this will include various laboratory and pilot plant test programs; and the appropriate analysis of the impact the various products may have on the environment for deposition;

    - Confirmation of process flowsheet required to achieve the proposed performance, which will complete the engineering requirements;

    - Development of the site infrastructure required to enable the permitting procedures - commencing with a new Environmental Impact Assessment (EIA) to be submitted to the relevant authorities; and

    - Confirmation of the initial assumptions to confirm the overall project requirements to a sufficient level of confidence such that the investment commitments can be made at the appropriate decision points.

25 Interpretations and Conclusions

The conclusions relevant to this study are:

25.1 Procedures

During site visits undertaken by Galen White, meetings were held with the database manager and procedures reviewed in the core yard, exploration department and SGS laboratory. Conclusions based on this site visit included:

- Procedures employed during logging, splitting and sampling of drill material is of an appropriate standard, with the core farm and digital data collection methods well managed.
- The onsite acQuire database is robust and of an appropriate standard however the historical database was not readily verifiable.
- SGS Assay laboratory in Kapan appeared to be well run, no formal audit was completed.

25.2 Geological Model

CSA believes that further work is needed to constrain the geological controls on mineralisation at Shahumyan.

The following aspects of the geological model require consideration:

- The current wireframes used in the MRE represent mineralisation, however, what part of the mineralisation is vein and what part is selvedge is uncertain.
- The potential for a discrete and potentially high grade gold mineralisation style has been observed by DPMK staff. This style is currently not being exploited and is characterised by subtle hydrothermal overprinting and argillic alteration (more typical of lode gold deposits). This style is in contrast to the dominant and visible quartz and carbonate veins with massive sulphides currently exploited.
- If this mineralisation exists at Shahumyan, it may explain the relatively poor correlation between lithology and grade.
- Some veins have been modelled dipping to the north. The controls that cause these veins to change dip are not clearly understood. It has also been observed by DPMK that most of the northerly dipping veins have better than average grades and larger than average widths.

25.3 Geological Logging

CSA considers the onsite procedures employed to collect and capture geological observations as appropriate.

However a review of the geological data collected showed a poor correlation between grade and rock type as well as density and rock type. This was demonstrated by:

- The prevalence of mineralised intercepts that fall outside of the vein model which have not been classified with a rock code indicating either a vein or a mineralised structure.
- The dominance of lithologies other than vein that are flagged within the Vein Model (predominantly andesite).

From both a resource as well as a mining perspective it is critical that controls on higher grades and densities are well understood. This has impacts on modelling and statistical domaining in future resource estimates as well as mining operations and development planning.

DPMK are currently undertaking a re-logging exercise which may help to address this issue. This may impact the model which is reliant on geological interpretations to establish continuity between mineralised intercepts.

25.4 Channel Sampling

Based on discussions with site, review work undertaken by CSA as well as observations made onsite made by CSA staff, it is believed that all drill and channel sampling data (including UC samples) are of sufficient quality and should be considered representative to be included in the estimation of resources.

25.5 Assay QA/QC

The results for blanks, standards, field duplicates and lab duplicates (repeats and splits) for gold, copper and zinc for the UG, SHA and UGC datasets have been reviewed and summarised below. All samples were analysed at the onsite SGS laboratory.

- CRMs and blanks performed acceptably.
- Field duplicates exhibited a significant bias to the original for gold, a lesser bias for Zn and no bias for copper. The mineralised body has a high nugget effect and as only a few field duplicates have been analysed, no conclusion regarding this bias can be made at this stage.
- Lab repeats have acceptable performance with no bias.
- Issues noted in the previous MRE (January 2013) such as instances of apparent misidentified QA/QC samples and anomalous blank results appear to have been addressed in this review period. Ongoing monitoring is required.

25.6 Database Validation

CSA undertook a suite of SQL validation checks on the DPMK database export provided and concluded that:

- The DPMK drilling data (surface and underground) are comprehensive, of a high standard and no significant issues were noted.

- Less data was available for the historical and channel datasets, but no fatal flaws were observed.

25.7 Historic Data Verification

A large proportion of the MRE dataset is reliant on historical data. This data has been digitised from hard copy plans and tables. Historical data also lacks QA/QC data as well as verification datasets regarding collar positions and down-hole surveying. As a result:

- CSA is unable to verify the accuracy or validity of the historical data against the more recent data collected by DPMK. In summary the grade populations correlate sufficiently to be included in the MRE.

- CSA understands that hard copy documentation of historic down-hole orientation data has recently been discovered by DPMK and is currently being entered into the database.

- DPMK located 563 hardcopy downhole survey reports in various conditions for the approx. 1,500 underground historic drillholes in the Shahumyan deposit. CSA believe that the new downhole survey data is an improvement relative to the previous dataset used in the MRE, although lack of initial azimuths at 0 m, lack of regular shots downhole and incomplete datasets are not an ideal.

25.8 Bulk Density

Conclusions from CSA's review of the density data include:

- Geological control regarding the distribution of densities at Kapan remains uncertain. This is largely attributable uncertain relationships between density values and geological logging of these samples for samples within and out of the vein model.

- Mine personnel are of the opinion that densities are higher within the vein than currently indicated by the data review, i.e. by flagging data by the model and/or classifying data based on lithology code.

- A clear trend between grade (Au Eq) and density supports this onsite observation; that mineralised veins are denser. As a result the use of a regression from which to calculate density based on estimated grades is considered appropriate. Based on this relationship a 4th order polynomial was defined and used in this updated estimate of resources resulting in an increase of 1% tonnes at a zero cut off (Au Eq) but with an increase of 1 to 2% tonnes above a 3 g/t Au Eq.

- Although the estimated densities reflect an improvement in modelling the variability of density in the block model, further work is required to improve the regression and to determine what density data is appropriate for use within the regression. This is made clear because the current dataset used for the regression (only data that was flagged by the vein model) does not reflect the grade distribution of the estimated model. Based on this discrepancy in distributions, it appears that the regression may be conservative in its calculation of density at lower grades.

- As additional density data is collected, the density assigned to the model may change and affect the resource tonnage.

25.9 Mineral Resource Estimation

158 vein wireframes representing 79 vein clusters were modelled by DPMK geologists, under the supervision of Mr Ross Overall, Resource Geologist DPM. Veins were interpreted using logged geology, underground mapping and drill hole structural data where available. Wireframes were interpreted on a nominal cut-off of 1.5 g/t Au Eq and 1 m down the hole length. Most veins were interpreted to strike approximately E-W/SE-NW and dip moderately to steeply to the south. Some veins were interpreted to dip to the north.

The vein wireframes were provided to CSA and used as the basis to create a set of mineralisation wireframes that honours the minimum mining width achieved at Shahumyan using the Long Hole Open Stoping method of mining. The CompSE process in Datamine software was used to selectively composite data to create the diluted wireframes. The composites produced had to meet a minimum grade of 1.5 g/t Au Eq and a minimum true width of 1.8 m (i.e. downhole lengths varied depending on orientation and dip of drilling, and orientation and dip of veins). 158 mineralisation wireframes were produced using Implicit Modelling and DTM Creation in Micromine software and were used to inform mineralisation volume in the MRE.

In addition to the mineralisation wireframes, a dyke model, development and mining depletion wireframes, and weathering surfaces provided by the client were used to flag data and the model.

Assay data was composited to 1.8 m true width to honour the dominant true width of vein intercepts. Statistical review of the resultant composites did not indicate bias. Data was composited using vein boundaries to start and stop compositing.

Statistical analysis was completed on the composites. Statistics for Au, Ag, Cu, Zn, Pb and S were reviewed. Au and Ag were characterised by heavy-tailed, skewed grade distributions, while the remaining variables had normal populations most easily interpreted using normal score transforms.

Variography was completed for use in Kriging. Variogram models for Cu, Pb, Zn and S were characterised by moderate to high nuggets for most variables, with two structures modelled. Gaussian transforms were used in the majority of cases and variogram models were back-transformed prior to use in Kriging.

The Top Cut Method for Deposits with Heavy-Tailed Grade Distributions (Rivoirard,2012) was used to estimate grades for Au and Ag. Cross variograms were produced for the truncated variable Au<6 g/t and the indicator for Au>6 g/t. Cross variograms were produced for the truncated variable Ag<110 g/t and the indicator for Ag>110 g/t. The variograms for the truncated variables were marked by low to moderate nuggets and longer ranges than might be achieved for the variogram of the untruncated variable. This is because the truncated variable has removed the noise / pure nugget effect associated with the high / extreme grades. The indicator variogram accounts for that part of the grade distribution associated with high and extreme grades.

Top-cuts were applied to Cu, Zn, Pb and S by reviewing histograms and probability plots.

Grade was estimated into a 3 m x 3 m x 3 m volume block model using Ordinary Kriging for Cu, Zn, Pb and S and the Top Cut Method for Au and Ag.

Swath plots were reviewed to assess semi-local scale reliability of blocks relative to input data along bench, easting and northing slices. Mean grades of inputs and outputs were compared and were within 2% of grades globally for Au, Ag and Cu. For Zn, Pb and S, there were larger differences with blocks reporting grades between 13% and 19% lower than input data. Histograms and probability of inputs and outputs were compared to assess level of smoothing. Visual validation of cross sections showed that blocks reflect the grade tenor of input data.

The estimation methodology used for Au and Ag in the vein model is believed to be appropriate because it reduces the influence of extreme values, while retaining the metal associated with them, and distributing it to the most appropriate blocks.

The MRE for the Shahumyan Deposit has been classified as Indicated and Inferred Resources based on the guidelines specified in the NI 43-101.

The MRE has been compared with that previously reported for the resource by CSA in July 2013 in “NI 43-101 Technical Report Shahumyan Project, Kapan, Republic of Armenia” dated 29 July 2013 and filed on SEDAR. The comparison is as follows:

- All mineralisation wireframes were created to honour a minimum true width of 1.8 m.
- A different AuEq equation was used to interpret the mineralisation and estimate grades.
- Underground channel sampling practises were verified and validated onsite and data from them were included in the current MRE (referred to as UC samples). UC samples were excluded from the previous MRE due to concerns over sample quality at the time of the site visit.

- A regression was used to estimate density from a well-defined relationship with AuEq grade, rather a single density average for all resources within the vein model.

- Mining depletion between 1 February 2013 and 31 December 2013.

- The current MRE is based entirely on modelled veins (wireframes). Approximately 34% of the tonnage and 28% of the Gold ounces from July 2013 MRE was estimated using probability methods.

The updated MRE has a 7% decrease in tonnes and a 3% increase in AuEq comprising 24% increase in Au grade, 4% increase in Ag grade, 7% increase in Cu grade and 26% decrease in Zn grade.

Of this, there was an 8% increase in Indicated tonnes, with a 3% increase in AuEq comprising 23% increase in Au grade, 16% decrease in Zn grade; 9% increase in Cu grade, and no change in Ag grade.

There was an 11% decrease in Inferred tonnes, with a 3% increase in AuEq comprising 23% increase in Au grade, 6% increase in Ag grade, 7% increase in Cu grade, 29% decrease in Cu grade.

25.10 Process Plant

- The mill has been upgraded over the past few years by the installation of new flotation cells. The crushing and milling circuits are from Soviet times, but are still considered to be in workable condition.
- The working conditions in the mill are good, with clear walkways, well-lit areas and uncluttered spaces.
- As part of the upgrade to 1 million tonne capacity, DPMK plans to refurbish three existing mills (one rod mill and two ball mills), a spiral classifier and crushed material bunker. They plan also to construct new mill feed system and a hydrocyclone battery, as well as to introduce several new flotation units to upgrade the capacity. The upgrade plan will effectively double the current grinding circuit capacity and will introduce the additional flotation capacity needed to process the increased throughput. The estimated cost of the upgrade program for the mill is $6 million as further detailed in Section 21. Another $13.4 million will be required to upgrade the existing tailings management facility to handle the increased volume of process tailings.
- A summary of the capital cost estimates is provided in Section 21 Capital Costs.

25.11 Mine Operations

Conclusions were drawn from observations made during the site visits by Julian Bennett and include:

- Mining equipment is generally in a good operational condition.

- Dilution factors at Shahumyan are very high, averaging 150%. There are a number of options to reduce this figure, including the use of a new drill rig. Mine management are fully aware of the need to improve this aspect of the operation.
- A new Simba drill rig has been introduced. This will assist the improvement of stope blasting which in turn should lead to better dilution factors. It will certainly increase the productivity of the mine operators.
- At the time of the first visit, capital development was beginning to be a constraint to the opening up of new production areas. To overcome this, a second Boomer rig has recently been introduced underground. There are skills issues that need to be overcome before the full utility of this new machine can be realised. During the second visit, some six months later, the situation was improving, and it is considered that the connection between the north and south sections of the mine will improve operations significantly.
- A single Boomer has the capacity to develop 150 m of capital development per month. So even with the new unit, it is considered that additional development equipment will be necessary over time.
- There has been a major effort to introduce standardised operational procedures to improve safety and productivity. It must be stressed that these two things go hand-in-hand to make the job easier, safer and better for everyone – operator and owner alike. This might not still be generally accepted.

25.12 Mine Stability

Conclusions were drawn from observations made during the site visits by Julian Bennett and include:

- No particular issues were noted during the site visit regarding mine stability and ground conditions underground were generally good.
- Some meshing is installed at major tunnel junctions. In the stopes, cable bolting is used to control the hanging wall of the stope in an effort to minimise dilution. It appears partially effective.
- The stopes are filled with mine development waste after completion. This is more an effort to get rid of the waste material underground rather than to provide mine support, although this latter is a secondary result and can only add to the general stability of the area.
- Ambient conditions underground were good, even where diesel equipment was working. Underground air control is with ventilation doors and booster fans.
- Compressors are kept in excellent condition and have reportedly performed well over the years.

25.13 Infrastructure

- Power is stated not to be a constraint at current mine production rates, and is not considered an issue.
- There is no evidence of water shortages.
- Road access is good however there is no direct connection to the national railroad system.
- There is some water seeping into the mine. This is handled appropriately with sumps and pumps underground and water is not an issue in relation to mining conditions.
- A number of self-contained refuge stations have been installed underground in central locations, providing safe refuge in the case of an incident. There are now two in the north section and two in the south.

25.14 Qualitative Risk Assessment

Table 69 presents the areas of uncertainty and risk associated with the project, and has been prepared from reviews completed by CSA, and informed by the conclusions summarised above, and recommendations discussed in Section 26.

Table 69. Project specific risks

Project Risk Area	Summary	Outcome	Mitigation	Qualitative Project Risk Rating
Mining	Planned and anticipated minimum mining widths built into the resource model may not be achievable.	Higher tonnage and lower grade mined, impacting operational profitability.	Stringent survey, grade control and reconciliation procedures required.	Moderate to High
Mining operations	New more powerful equipment is being introduced, and production pressure will increase.	There is a possibility of accidents and incidents.	More intensive hands-on training, the continued use of and adherence to defined standard procedures.	Moderate to High
Geology	Structural controls on vein dip are not clearly understood.	MRE classification upgrade barrier.	Structural analysis.	Moderate
Geology	Geological interpretation from core logging, geological coding	Limited understanding of higher grade geology/altera-tion association.	Training of geologists to provide standardised and relevant logging.	Moderate

Project Risk Area	Summary	Outcome	Mitigation	Qualitative Project Risk Rating
	masks mineralisation association.		Review logging procedures to ensure they are fit for purpose.	
Geology	Underground channel sampling procedure not adequate.	Sampling bias, overstating of grade.	Implement improved procedure (underway). Provide training to samplers and geologists. Ensure procedures are being. followed.	Moderate
Resources	Vein zone wireframes that require adjustment as part of future MRE updates, may have portions that are below the minimum mining width.	Possible changes in tonnage/grade.	Wireframe editing to include mining criteria such as mining widths and grades into the vein interpretation.	Moderate
Resources	Probabilistic Modelling not fully validated by drill testing or field checking.	Resource classification upgrade barrier.	Drill testing and field checking as part of future resource development works.	Moderate
Data Verification	Limited information available with respect to historic data collection.	MRE classification barrier.	Source additional information where available (underway). Include in future MRE.	Moderate
Reconciliation	Issues identified in current reconciliation include: reporting regime in GEMS overstating volume; milled tonnes don’t relate fully to mined out volumes because material from other parts of the mine are included; impact of	Uncertainty over performance of resource model versus production due to sub-optimal reconciliation.	Additional software training of personnel involved in reporting may be required to ensure correct volumes and grade are reported. Identify and control what material is getting reporting - e.g.	Moderate

Project Risk Area	Summary	Outcome	Mitigation	Qualitative Project Risk Rating
	stockpile feed not understood.		other parts of the mine, stockpile feed etc. Review practices.	
Process plant upgrade	Majority of equipment is already existing (mills, spiral classifier, bunker) and the upgrade requires maintenance related activities. The new equipment to be purchased (feeders, hydrocyclone, flotation cells) are sized by similitude to the existing operational units.	There is a possibility of accidents and incidents during construction	More intensive hands-on training, the continued use of and adherence to defined standard procedures.	Moderate

26 Recommendations

The recommendations relevant to this study are:

26.1 Geological Model

CSA recommends that results from a re-logging program (which was underway at Shahumyan in 2013) should be reviewed:

- It is recommended that more detail about veins be logged and that this information be used in the construction of wireframes representing vein mineralisation alongside existing wireframes that represent vein and selvedge mineralisation.
- To identify potential mineralised shear zones which have been observed, but about which there is uncertainty about continuity or structural control.
- To improve confidence regarding continuity of interpreted vein models.

26.2 Geological Logging

Regarding current DPMK geological logging, CSA recommends:

- Logging of core to 1 m intervals should be replaced with logging honouring geological boundaries.
- Logging to have more detail regarding vein, percentage of vein and other salient detail required to adequately model that mineralisation associated with vein.
- Lithological logging should be better placed to indicate mineralisation and confirm and support continuity of grades between drill holes, which is currently informed predominantly by grade intercepts.
- Training should be provided to on-site geologists to assist identification of mineralisation outside known vein sets as well as increase the amount of detail collected through logging to assist with geological modelling.
- A review of logging should be completed to ensure that all relevant data is recorded in the database that gets used in the interpretation of mineralisation and the MRE.
- Geologists who undertaken geological logging should constantly review logging against assay results as field verification and training.
- Efforts should be made to include vein number as part of the logging process.
- Include geotechnical logging as part of the process to provide additional data for mine stability analysis.

26.3 Assay QA/QC

No fatal flaws were noted in either the database review or the QA/QC review. Recommendations include:

- Review of field duplicates should continue in order to determine the correlation between original and duplicate results. A bias was noted in the gold field duplicates, which requires ongoing monitoring (only a small sample population was analysed).
- Currently, QA/QC is monitored on a batch by batch basis. QA/QC should be reviewed over longer periods to assist with noting trends in the results (such as analytical drift).
- Umpire 'third party' assaying at an accredited laboratory is recommended as part of a comprehensive QA/QC procedure. QA/QC material should be included with the umpire samples.

26.4 Bulk Density

CSA recommends a detailed study to constrain the geological controls on density. At present there is little correlation between density and lithological groupings which should be reviewed.

This study could include a program of channel sampling across clearly defined vein exposures and across entire stopes, would be beneficial in understanding the impact of small density variations across a mined width. Incorporating this with photos, mapping and detailed lithological coding would be beneficial with the objective of verifying the regression used and/or having sufficient data within all major vein groups to rely on flagging of density data. Bulks samples if feasible may also be an appropriate option.

26.5 Mineral Resource Estimation

CSA recommends that a review of wireframes be completed to ensure modelled thicknesses using the new wireframe process are honouring the minimum mining width as a minimum.

Narrow vein deposits are often better estimated using 2D methods (grade, thickness). However, there are too many veins at Shahumyan for this to be a practical solution. It is recommended that the current 3D model be compared against a 2D estimate of one of the major veins, in order to test the performance of the current model and to help to refine parameters in future resource estimates.

Investigation of the understating of zinc grades is recommended prior to the next MRE update. This should include, but not be limited to, a review of top cutting to ensure top cuts imposed are not too harsh.

Results from the grade control and reconciliation processes currently underway should be fed back into the MRE to assist with continual improvement.

26.6 Grade Control and Reconciliation

Reconciliation of the resource model and grade control models against mined is strongly recommended to improve the confidence in the model. CSA is currently assisting DPMK with a program to develop a grade control model and implement important reconciliation procedures. The results should be reviewed on an ongoing basis, and results integrated into the next resource update as necessary.

26.7 Operations 2014

CSA have reviewed the operation plans as outlined by DPMK and considered them to be appropriate.

26.7.1 Near-mine Development

Near-mine exploration work planned for 2014 is primarily focused on completion of the 2013 near-mine drilling program. Once results have been interpreted, future drilling may occur later in 2014 which will target encouraging intersections and potential extensions to the current mineralised body.

26.7.2 Regional Exploration

Regional exploration soil sampling field programs are expected to commence in March 2014, focusing on covering the remaining area within the exposed Middle Jurassic sequence that was not covered during 2013, as well as infill and detailed sampling programs around anomalies identified from the 2013 soil sampling campaign, focusing on the Centralni South and Antarashat South areas. Additional plans for 2014 are:

- Detailed geological mapping of soil anomalies identified during the 2013 soils campaign
- Trenching of soil anomalies
- Based on soil and trenching results; tentative drill testing of encouraging prospects is planned for Q3 2014.

26.8 Mining

The mine is undergoing a series of upgrades, including the purchase of new more powerful equipment. Production is slated to increase so that from year 3, the mine will be producing at an annual rate of 1.0 Mtpa. CSA recommends that the training of personnel is critical to facilitate these upgrades.

The current introduction of improved operational procedures will be key to this improvement. Standardised operational procedures are sometimes not seen to be relevant to an underground operation. This view must be counteracted at all levels in the organisation. The message must be got over that productivity and safety go hand-in-hand, and that everyone in the organisation must expect to go home safely each and every day. There are so many examples around the world that safe mines with good operational procedures and standards are also the most profitable and productive.

The recent push to improve the adoption of standard procedures to boost safety and efficiency is applauded. However, standards can easily slip, and management at the mine at all levels must constantly be aware that the only way to reach acceptable standards is to reinforce the need for compliance at all times. The reasons for compliance must also be driven home.

The high level of ROM dilution is recognised and better stope drilling equipment is being introduced. But this alone will not fully address the dilution factor.

CSA recommends that a serious technical review be made of mining methods in order to reduce the quantity of mine waste in the ROM stream and to improve productivity.

CSA notes that the improvement in the quality of management is noticeable and it is reasonable to expect that these various issues will be improved on over time.

26.9 Process Plant

As part of the upgrade to 1 million tonne capacity, DPMK plans to refurbish three existing mills (one rod mill and two ball mills), a spiral classifier and crushed material bunker. The plan also to construct new mill feed system and a hydrocyclone battery, as well as to introduce several new flotation units to upgrade the capacity. The upgrade plan will effectively double the current grinding circuit capacity and will introduce the additional flotation capacity needed to process the increased throughput. The estimated cost of the upgrade program for the mill is $6 million as further detailed in Section 21. Another $13.4 million will be required to upgrade the existing tailings management facility to handle the increased volume of process tailings.

A detailed estimate of those capital costs is presented in Section 21 Capital and Operating Costs.

26.10 Operations 2014

26.10.1 Operations at Shahumyan

- During 2014, approximately 32,000m of underground drilling with an additional 12,600m of surface drilling. The scope of this drilling is to increase resource confidence in planned production areas for mine planning purposes. Three underground and two surface diamond drill rigs are currently working to meet this target.

- 9,510m of drilling is planned in the central section to delineate vein zones 22, 30, 34, 102, 104 and 105 for mine planning purposes.

- 27,470m of drilling in the north section to define veins zones 3n, 5n, 9, 13, 32, 35, 46 and 103 for mine planning and to re-categorise resources.

- 7,640m of drilling in the south section to for veins zones 5s, 26, 27, 28, 29 and 31 for mine planning and to re-categorise resources.
- DPM is committed to spend $2,400,000 USD for underground drilling and $885,000 USD for surface drilling.

26.10.2 Near-mine development

- Surface drilling within the early portion of 2014 will focus on testing the possibility of extensions of the Shahumyan deposit within poorly explored areas of the near mine region

During 2014 follow up work is planned to further understand the fault offsets in the mine area via detailed surface mapping and interpretations of UG and surface exposures

26.10.3 Regional Exploration

- Regional exploration programs will involve prospecting and 1:2000 scale geological mapping of soil geochemical anomalies defined during 2013.
- Any proposed diamond drilling is contingent on the success of the target build-up work.

27 References

Armenian Ministry of Natural Resources, 2012, Licence Document 29/183 – 27th November 2012 – Armenian Ministry of Natural Resources

Armenian Ministry of Natural Resources, 2012, Mine Claim Permit 29/183 – 27th November 2012 – Armenian Ministry of Natural Resources

Armenian Ministry of Natural Resources, 2012, Mine Claim Contract (Appendix N1) – 27TH November 2012 - Armenian Ministry of Natural Resources

Cambourne School of Mines Associates (CSMA), 2003 – "Report on testing of 2 future ore Samples from the Shahumyan Mine"

CSA, 2013[1], Site visit to Shahumyan Project, Kapan, Armenia, for Dundee Precious Metals, undertaken between 5th February and 7th February 2013, Author: White, G

CSA, 2013[2], Mineral Resource Estimate (dated January 2013), NI43-101 Report, For Dundee Precious Metals Inc., Shahumyan Project, Kapan, Republic of Armenia, dated 29 July 2013, Authors: White, G.W., Titley, M., Bennett, J., O'Connor, M

CSA, 2014, Site Visit, Grade Control Review, Kapan Mine, Armenia, Author: Rose, S, dated 18 February 2014

Davis, K.B., 2006, Report on the Geology of the Kapan Area with a Focus on the Shahumyan Deposit, observations and interpretations based on a visit in September 2006, for Dundee Precious Metals Inc.

Deno Gold. 2004. Kapan Ore Mining and Processing Enterprise Operations Summary As At July 2004. Internal document

Deno Gold Mining Company (GMC), 2010, Annual Interim report on geological exploration of the ore fields and supplementary exploration of Shahumyan gold-polymetallic deposit, prepared by Deno GMC

Deno Gold Mining Company (GMC), 2011, Annual Interim report on geological exploration of the ore fields and supplementary exploration of Shahumyan gold-polymetallic deposit, prepared by Deno GMC, Kapan 2011, authors: Harutunyan, K. and Navasardyan, S

Deno Gold, 2006 – Centralni Mine Operating Records

Dubchak. P., 2009. Rare Metals Accounting in the Shahumyan Deposit (Cd, In, Ga, Te, also Se). MEMO

Dundee Precious Metals Kapan, 2014 – Operations Metallurgical Balances

Dundee Precious Metals, Toronto, 2012, 2013 – Annual Sustainability Reports

Dundee Precious Metals, Project Group, 2014 – Capital Estimate for DPMK Expansion to 1 Mtpa (20140825).

Geomap, 2012, Review of Geological Mapping of the Kapan Area, Armenia, prepared for Dundee Precious Metals Inc., November 2012, author: N.M. Tate

Golder Associates (Canada), The Mining and Metallurgical Institute (Armenia), Kingett Mitchell (New Zealand), AATA International (USA), Geolink Consulting (Russia), Combined Report 2008 – Environmental and Socio-Economic Baseline Study.

Golder Associates, Report 1 - Geghanush Tailings Management Facility Closure Study (Golder Report No. 12514150374.501/B.0), and

Golder associates, Report 2 - Northern Tailings Management Facility Pre-Feasibility Study (Golder Report No. 12514150374.502/B.0).

Golder Associates, Sudbury, Canada, 2011 – "Laboratory report on Deno Gold tailings", Report # 11 – 1900 – 1004.

Golder Associates, March 2013 – "Tailings Management Facility – Prefeasibility Study". Report # 1251450374 500/B.0

Golder Associates, December 2013 – "Geganush Tailings Management Facility Kapan – High Level review of the viability, costs and considerations of extending the life of the TMF" Technical Memorandum, #12514150607/TMF

Jigsaw, 2007, Observations and Interpretations from Mapping in the Kapan region, Armenia, prepared for Dundee Precious Metals, October 2007, authors: Wood, D., Noble, M., Standing, J. Outhwaite, M.

Jigsaw, 2011, Report on Mapping and Interpretation of the Kapan Region, Syunik Province, Armenia, prepared for Deno Gold Mining Company, January 2011, authors: Standing, J., Outhwaite, M. and McCormack, B.

1 Mederer, J., Moritz, R., Zohrabyan, S., Vardanyan, A., Melkonya, R., Ulianov, A., Spikings, R. & Selby, D., 2-12, 2012, Geology, Mineralogy and Geochemistry of Polymetallic and Copper-Pyrite deposits of the Mesozoic Island Arc in the Lesser Caucasus, European Mineralogical Conference 2012 Poster

2 Mederer, J., Moritz, R., Ulianov, A., Spikings, R. & Selby, D., 2012, The Kapan mining district in the Lesser Caucasus: multiple ore-forming events during the northeastwards subduction of the Tethys below the Eurasian margin, European Mineralogical Conference 2012 Abstract

SGS Lakefield (Canada), 2012 – "An Investigation by High Definition Mineralogy into The Gold Mineralogy of the Deno Copper Concentrate"

SGS Lakefield (Canada), February 2011 – "QEMSCAN Data for (2 Deno Gold, and 1 Chelopech Concentrates) for DPM" Project CAVM – 50101 – 001, M17000 – Jan 11.

SGS Lakefield (Canada), 2011 – "An investigation into the Gold, Silver and Tellurium deportment in Zinc concentrate (from Deno Gold)", Final Report Project 12613 – 101, MI 5004 April 2011.

Sloan, R., 2006. Dundee Precious Metals – Project Rush Grinding and Floatation Test Work Report. SGS Minerals, Toronto.

Spectrum Survey and Mapping, 2006, Deno Gold Mining – Kapan Operations, Report on the GPS implementation and survey status – 5th October 2006, author: Steven Glavinas, Authorised Mine Surveyor, WA

Tate, N.M., 2012, Review of Geological Mapping of the Kapan Area, Armenia, Report compiled by GEOMAP for Dundee Precious Metals Inc., November 2012

Vardanyan, V., 2014. DPMK Internal Review of the 2008 Baseline Study (Golder et al) – September 2014